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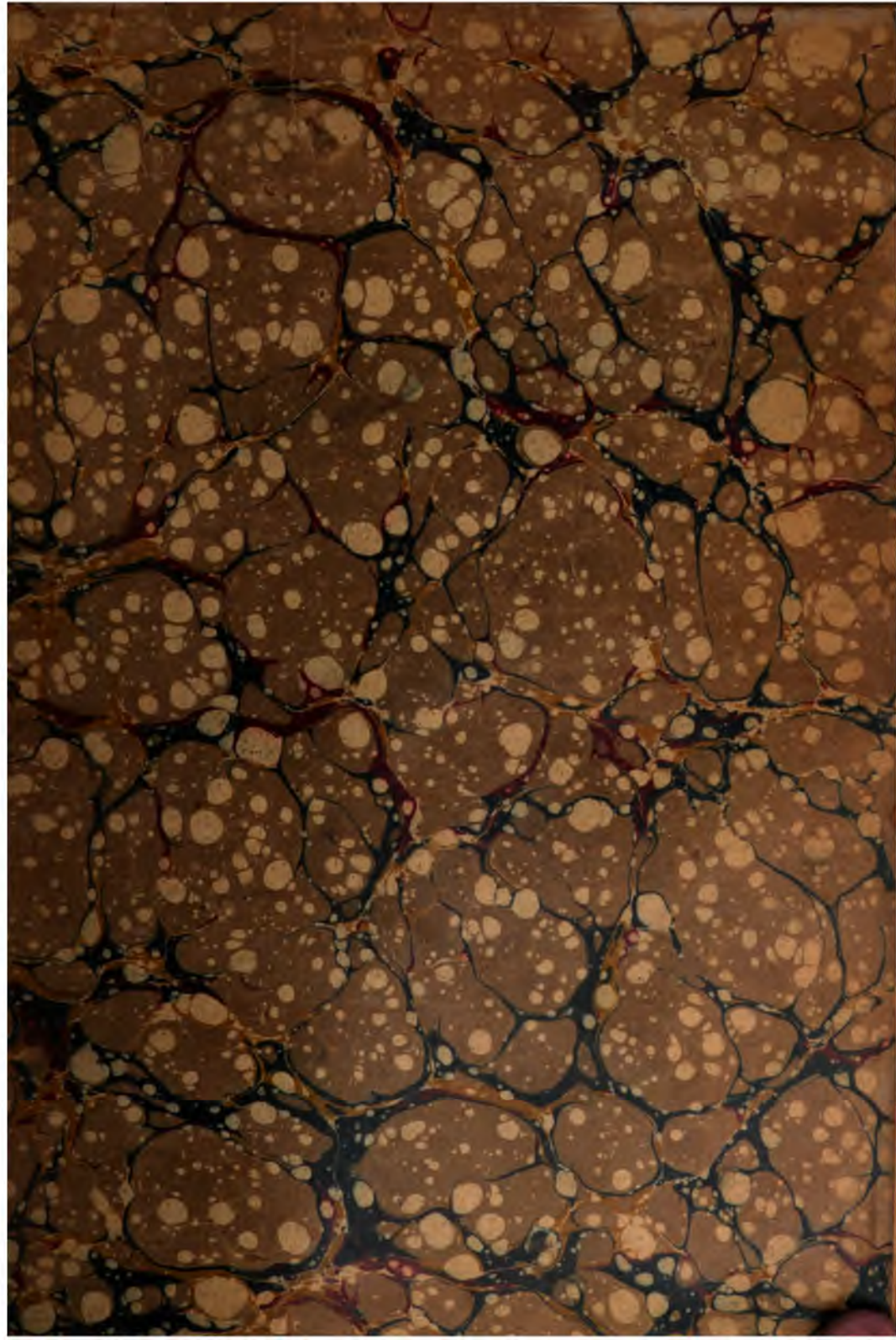
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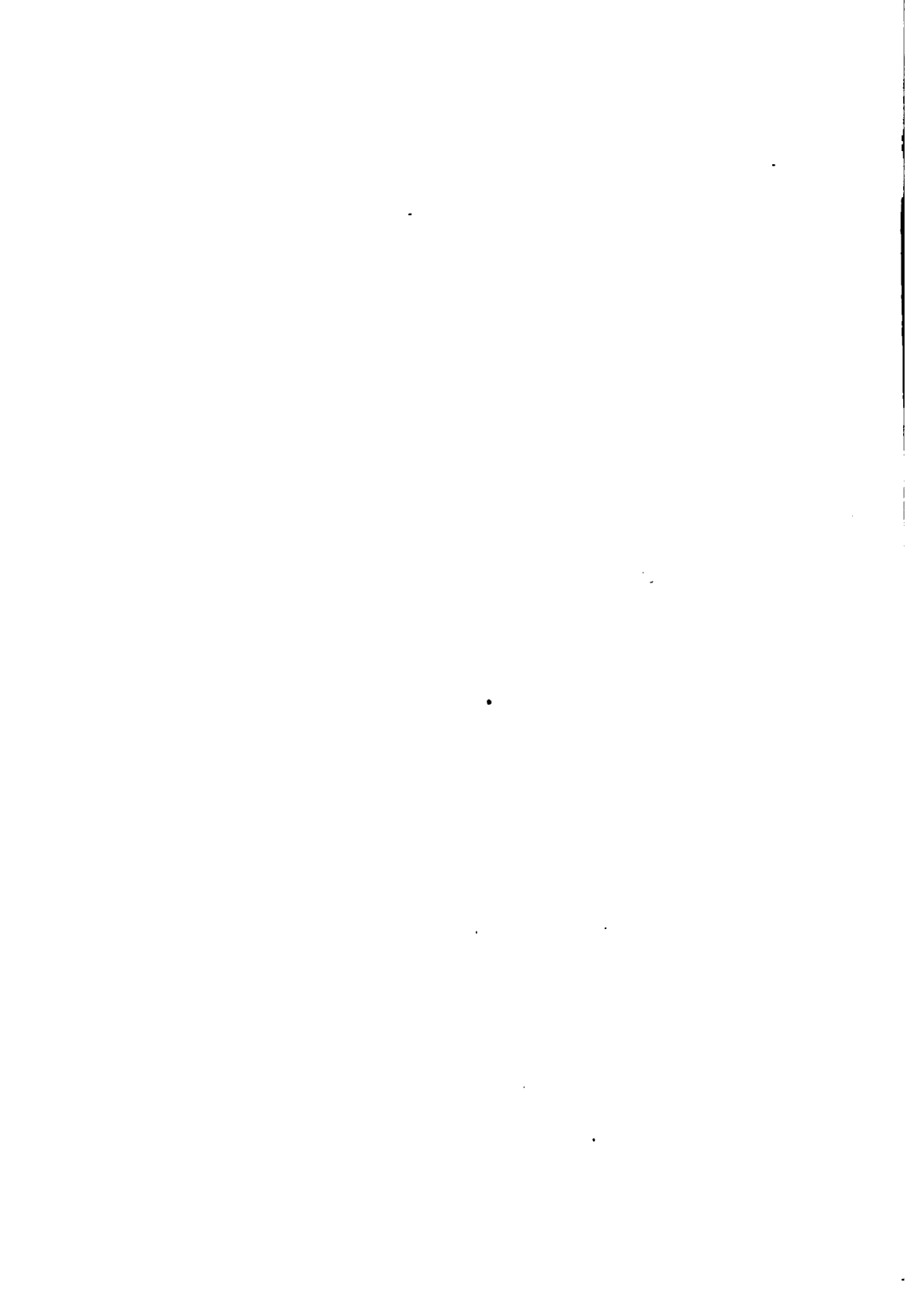
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than is usual, but it often happens that it is not the highest extraction, nor yet the most perfect design of plant, which results in the greatest nett profit in the aggregate. These are points which cannot be too carefully considered, and in trying to impress the reader with this side of the subject, we have dealt with the matter of costs rather more fully than might at first sight appear necessary.

We have attempted to interest the practical worker in fundamental principles, with the object of inducing him to improve and develop the process, and to some the discussions may, at first, appear too theoretical for a book of this character. But everyone who tries to improve, acts on a theory of some sort, though not always conscious of it, and on the soundness of this theory success largely depends. A knowledge of theoretical principles is therefore beneficial, if only in conducing to prevent fallacious reasoning.

The manager of a cyanide works is constantly asked to decide on the adoption, or otherwise, of a so-called improvement; and in order to be able to discriminate successfully between the real and the apparent, it is necessary to use sound reasoning, based on theoretical principles, and supported by practical experience. For, to make an alteration without investigating the reason that leads up to it, is little short of gambling; or to adopt a so-called improvement only because it has been successful elsewhere, points to incompetency.

The chemistry of the process being chiefly of a physical nature, we have employed the newer developments in physical chemistry to explain changes that occur in dissolution and precipitation. The application of these modern developments are, we believe, likely to become of far-reaching importance, not only in giving a deeper insight into unsolved problems in the process, but in helping to overcome difficulties that have hitherto appeared insurmountable.

The views put forward regarding dissolution and precipitation are based on experimental evidence. The methods of conducting the experiments and the proofs of the results would be out of place in a book of this sort, and for this reason we simply put forward the views as statements of facts proved only to our own satisfaction. The comparative results given on the dissolution of metals and minerals were obtained from average samples, and are the average of several determinations. This is necessary, because different samples of the same metal or mineral often gave different results; and further, because irregular variations occurred, in the dissolving effect of the solution, at different stages of the investigations; even when extreme care

was taken in determining the percentage of cyanide present, and in maintaining the same temperature. The values given should therefore be taken as preliminary, pending further investigations, but for practical purposes they may be regarded as sufficiently correct. In some cases, to save time, the results were obtained by interpolation from empirical formulæ, as this method was found to agree with direct experiment.

We have purposely avoided any abstruse mathematical reasoning, as we are appealing more to those whose knowledge of figures is limited, and therefore we have gone but little beyond the rudiments of arithmetic and algebra. Undoubtedly even this extent will be found greater than a certain section of those engaged in the industry command, but we do not think that that is a sufficient reason for omitting what to others may be helpful. And since this book is intended not only for the shiftman, but for the engineer and manager, who, it is presumed, are acquainted with, at least, this slight knowledge of figures, we need say no more.

The formulæ and rules are, for the most part, deduced from practice. Constants are used which may be adjusted to meet abnormal conditions, and thus the formulæ can be universally applied. Copious tables are given, as the best means of simplifying calculations and of expressing results for practical men.

Although great pains have been taken to secure accuracy in the calculations, yet in a book like this, where there are so many figures involved, it is probable that errors have crept in. We should therefore be very grateful to those who would bring to our notice any mistakes.

The illustrations are chiefly from working drawings to scale, reduced, which have been, for the most part, largely used in practice, and sufficient dimensions are given for an engineer to work from.

Much care has been taken in quoting the sources of our information, and we have endeavoured to give full credit where it is due. In some cases, however, a doubt exists as to whom the credit belongs, and then it has been given to the earliest one found published, but in a few cases we were obliged to omit names altogether for want of more definite particulars.

Besides those whose names are mentioned in the text, we are specially indebted to Mr E. P. Martin, B.C.E., New Loch Fyne mine, Victoria, and Mr G. Kermode, B.C.E., Castlemaine, Victoria, for Australian prices; to Mr Charles Butters, A.M.I.C.E., of London, who has placed at our disposal valuable tabulated results of experiments made on a working scale by his staff in America; to Mr Ernest Williams, mining engineer, Johannesburg, Mr John

Kelly, consulting engineer, Johannesburg, Mr Clement Dixon, Johannesburg, Mr W. Watkins, A.M.I.C.E., London, Mr M. Torrenti, Johannesburg, and Mr Francis Spencer, Johannesburg, for general technical information.

Our thanks are also due to those secretaries of companies who have shown us much courtesy and given assistance in obtaining costs, etc., amongst whom are Mr J. N. C. Humphreys, secretary to Charles Butters & Co. ; Mr Herbert Akers, London, secretary of the Waihi Gold Mining Company ; and others whose names are mentioned in the text.

Much as has been done during the past decade in improving the process for treating simple ores and lowering costs, there is still much left to be done before we shall be able to cope with the problem of cyaniding the large bodies of complex ores known to exist. This end will, in our opinion, be most quickly arrived at by constantly bringing together, in a concise form, exact data of researches made under different conditions, and thus facilitating development in the right direction. With this object in view, we earnestly appeal to all connected with the process for short particulars of investigations or working costs which they may wish to submit to us for recording, together with the source of original publication.

In bringing together this fragmentary information concerning the cyaniding of gold and silver ores, we are aware of having fallen far short of our aim, but we trust that this small contribution may be of some use as a guide, and may serve to increase that interest in the process already so strongly manifest in many parts of the world.

H. F. JULIAN.

EDGAR SMART.

REDHOLME, TORQUAY, ENGLAND.  
P.O. Box 830, JOHANNESBURG, S. AFRICA.  
*December 1903.*

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# CYANIDING GOLD AND SILVER ORES.

## CHAPTER I.

### EARLY HISTORY OF THE CYANIDE PROCESS.

ABOUT the beginning of last century it was first notified that cyanide of potassium had a solvent action on gold. No practical use seems to have been made of this fact until towards the middle of the century, when Elkington patented his process for electro-plating articles with gold and silver from cyanide solutions. Some interesting investigations were made soon after this by Bragation,\* Napier,† Elsner,‡ and Faraday.§ These investigations were chiefly of scientific interest, and no suggestion referring to the dissolving of the gold out of ores appears to have been recorded until, in 1866, Wurtz|| mentioned it in a paragraph of a letter. The following year a patent was taken out in the United States by J. H. Rae, entitled an "improved method of treating auriferous and argentiferous ores." The process consisted, in the main, in agitating the crushed ore in a solution of potassium cyanide and simultaneously passing an electric current through the solution to facilitate the dissolving of the gold and silver, and at the same time to precipitate the dissolved metals on cathode surfaces of copper. It seems probable that the use of a cyanide solution to dissolve gold and silver from ores was, to Rae, common property at the time, and that the employment of the electric current was the chief novelty. It is reported that Rae's process was tried on a commercial scale, but with what success is not stated.

Between the granting of Rae's patent and the year 1885 several patents were applied for in which cyanide was used in such a way that the gold and silver in ores was caused to dissolve, but none of these inventions were of importance. During that period a number of investigations had been recorded on the dissolution of gold in ores by means of cyanide. Hahn¶ in 1870 stated that the gold present in sulphides in ores is soluble in a cyanide solution, and

\* *Bull. de l'Acad. des Science St. Pétersbourg*, 1843.

† *Phil. Mag.*, vol. xxv. p. 64, 1844.

‡ *Érém. Journ. Prakt. Chem.*, vol. xxxvii. pp. 441 to 446, 1846.

§ *Roy. Inst. Proc.*, vol. ii. p. 808, 1857.

|| *Am. Journ. Science*, vol. xli. p. 222, 1866.

¶ *Chem. Centralblatt*, No. 15.

he pointed out that this fact was taken advantage of in the treating of gold ores. W. Skey\* in 1875 made some observations on the action of potassium cyanide on gold ores as then employed in the amalgamation process. He pointed out that the use of potassium cyanide in connection with amalgamation occasioned a loss of gold and silver, as these metals were soluble in the salt. He asserted that both gold and silver in a cyanide solution were electro-positive to mercury, iron, and metallic sulphides found in the ore, although the latter metals and sulphides were acted on at the same time, but to a lesser extent. That is equivalent to saying that the gold and silver would become dissolved out, while the sulphides would be little acted on.

It was common practice at that time amongst mill men to treat concentrates for their gold by barrel amalgamation, and it was known that in order to render the gold readily amalgamable the addition of some potassium cyanide had a very beneficial effect. Skey observed that the use of cyanide for this purpose was attended by a loss of gold and silver, as these metals dissolved and were not again precipitated by the mercury or the sulphides, the latter being electro-negative to the gold and silver; but he proposed that in order to avoid loss from this source the fluid used in the barrel should be passed over copper plates. The copper being electro-positive to gold and silver in cyanide solutions, precipitates these metals.

W. A. Dixon, in a lecture delivered before the Royal Society of New South Wales in 1887, alludes to a patent that had been taken out in America for extracting gold from its ores by dissolving the metal out with potassium cyanide, and states that he does not think the process likely to prove satisfactory on account of the cost, the instability and poisonous nature of the salt. These drawbacks had doubtless presented themselves to many minds.

In the year 1885 J. W. Simpson applied for a patent in the United States for a process of separating gold, silver, and copper from their ores, and this consisted in subjecting the ore to a mixed solution of potassium cyanide and carbonate of ammonia, and subsequently precipitating the dissolved metals by means of zinc plates. He points out that his process is particularly applicable to sulphide ores that have been finely pulverised. To remove the precipitated metals from the zinc plates, he suggests agitating them in sulphuric or hydrochloric acid.

It would appear that Simpson added carbonate of ammonia, with the idea that it could be used to facilitate the dissolving of the gold in place of the electric current used by Rae, for he says in his original specification "that by the use of carbonate of ammonia the necessity of electricity is obviated, and the cost of separating the metal from its ores is greatly reduced." He must, however, have found this to be an error, as in the final specification the statement was suppressed. It does not appear that Simpson ever applied this process on a commercial scale.

About the time that Simpson was applying for his patent various metallurgists began to interest themselves in the cyanide treatment of gold ores, and

\* *Trans. and Proceed. New Zealand Inst.*, 1875.

recorded some results. Among these were Endlich and Mühlenberg, W. A. Dixon and L. Janin, jr.

In the year 1887 Wanliss and Julian, then acting as consulting engineers to the Johannesburg Pioneer Company, erected a small plant to treat blanketings from the company's first stamp mill. It consisted of an amalgamation barrel, a receiver for the treated ore, and an amalgamated copper plate with mercury wells. The barrel was charged with about  $\frac{3}{4}$  ton of blanketings and one-fourth to one-third their weight of water. To this was added about  $\frac{3}{4}$  lb. of 60 per cent cyanide and 1 lb. of caustic soda. The barrel was then rotated for eight hours, when it was opened and a bottle of mercury (72 lbs.), in which  $\frac{1}{2}$  lb. of commercial sodium amalgam had been dissolved, was added. The barrel was again rotated for eight hours and the contents were discharged into a receiver, from which it was slowly washed out with water to collect the particles of amalgam on the plate and in the mercury wells. The gold extracted by this little plant amounted to about 90 ozs. per month.

The assays showed that a considerable amount of gold still remained in the tailings undissolved, and it was found impracticable to get anything like a complete precipitation of the dissolved gold in the time allowed. A longer treatment might have been given and a better extraction obtained, but that meant a decreased output of gold, which is a serious item to a small struggling company just beginning operations.

A much more perfect plant was erected at the Roodepoort United Main Reef Company's mill in 1888, which consisted of two barrels, each capable of treating  $1\frac{1}{2}$  tons of blanketings at a time in the same way as at the Pioneer Company, except that the pulp was heated with live steam until it was just too hot to bear the hand in. The ore in this case was less than half as rich as that treated at the Pioneer Company, but the dissolving of the gold was so complete that the washed residue tailings rarely assayed as much as 1 dwt., while the originals were sometimes  $1\frac{1}{4}$  ozs. The difficulty of dissolving the gold was thus got over, but the actual gold recovered did not at all tally with the assays, there being a discrepancy, which was found to be due to bad precipitation by the sodium amalgam. About this time Julian was making investigations in connection with the dissolution of gold in ores by chlorine, and also by cyanide under a high pressure of air, and subsequent precipitation by an electric current and by sodium amalgam, which resulted in some patent applications in South Africa. For oxidised ores the dissolution of the gold by chlorine under pressure of air and precipitation by sodium amalgam was as perfect as could be desired; in fact, some blanketings containing 34 ozs. per ton left only about 7 dwts. in the residue, equivalent to a 99 per cent. extraction. The fine gold was dissolved and re-precipitated as amalgam, while the coarse gold was made easily amalgamable. But when pyritic ore was dealt with, the dissolution of the gold became impracticable without an excessive consumption of chlorine, and it was found necessary to again return to cyanide as a solvent, and in order to precipitate the dissolved metals an electric current was employed, the anode being iron or lead



peroxide, and the cathode mercury. A number of experiments had been carried on at Johannesburg and Kimberley in 1888-9 embodying these latter ideas, dealing with ton lots at a time.

In 1889 a plant was erected at the Wemmer Gold Mining Company, Witwatersrand, to deal, by this process, with ore that carried some pyrites. It consisted of three rotating barrels, each having a capacity of two tons of ore. About the same proportion of cyanide and alkali was added as in the previous cases, but a pressure of air was kept up in the barrels of about 30 to 40 lbs. per inch, and the contents were heated with live steam. The results showed that when the ore was not too coarsely crushed the gold dissolved rapidly and very completely within a very short time. When practically all the fine gold was dissolved the pulp was discharged into a circular agitating vat of wood. The bottom of the vat was covered with a well amalgamated copper plate, coated with an excess of mercury. This formed the cathode of an electrolytic cell. The anode consisted of an iron disc stirrer which kept the ore particles in a state of suspension and continually brought fresh solution in contact with the mercury cathode. The side of the vat was also lined with iron to within a few inches of the bottom and connected to form an anode. After the pulp had been agitated and the electric current passed through it for some hours, the vat was slowly discharged through sluice gates arranged at different levels, with the addition of more solution. The pulp then flowed over well 'set' amalgamated plates and through mercury wells or riffles. The baffle boards of the riffles were covered with lead peroxide and connected with the dynamo to form anodes, while the mercury and plates were the cathodes. The mercury cathode in the vat soon became sodium amalgam. This was removed at intervals and added to the ore in the barrels to assist precipitation.

It was found that the electrode surface was too small to get anything like complete precipitation in the longest time that could be practically allowed, and to partly compensate for this the solution was circulated and kept continuously coming in contact with the electrodes. The amalgam obtained, either by precipitation with sodium amalgam or by the electric current, was of a fine slimy nature and could not be squeezed, but if allowed to stand in a pail for a few days it assumed, to some extent, the characteristics of ordinary amalgam.

At the time that the small plant erected by Wanliiss and Julian was being worked by the Pioneer Company, J. S. MacArthur, R. W. Forrest, and W. Forrest, who had been experimenting in Scotland with samples of gold ore, applied for a patent in Great Britain, No. 14,174, 19th October 1887, for extracting gold and silver from ores by dissolving out the metals with a solution containing cyanogen or a cyanide. At a later date they state, in the United States application for patent, that: a solution containing from two to eight parts by weight of cyanogen per 1000 of water without the presence of any other chemically active agent "has a selective action such as to dissolve the gold and silver in preference to the base metals." They suggest that the

gold and silver may be obtained from the solution by any convenient known way, "such as evaporating the solution to dryness and fusing the resulting saline residue, or by treating the solution with a sodium amalgam."

It is probable that this patent would never have been noticed by practical men were it not that MacArthur and Forrest followed it up with an ingenious method of precipitating the dissolved metals by zinc. The fact that gold and silver could be dissolved from ores by a cyanide solution did not add anything to the knowledge previously possessed by metallurgists, and the discovery of the so-called 'selective action' of weak solutions is a factor of no practical importance. This is self-evident from the fact that other factors than strength of solution, viz., time and cost of material, have to be taken into account, so that the most suitable strength of solution is often not within the limits of the solutions said to have selective action.

MacArthur and Forrest added, to the gold-mining industry, knowledge of great value, which has been highly appreciated by practical men, through the introduction of zinc in the form of fine shavings or turnings for precipitating gold and silver from cyanide solutions. They covered this invention by a patent obtained in Great Britain, No. 10,223, and dated 14th July 1888, of which the second claim consisted "in precipitating gold and silver from cyanide, chloride, bromide, thiosulphate, sulphate, or other similar solutions, by means of zinc, the employment of the zinc as freshly prepared in a state of fine division." The claim has been considered too broad, inasmuch as in the years 1862-4 \* it was pointed out that zinc in a granular condition is particularly suitable for precipitating from solutions most metals occurring in ores; also, a patent was obtained in Great Britain in 1884 by A. P. Price for precipitating gold and silver from solutions resulting from the treatment of ores by agitating the solution in contact with zinc in a fine state of division. Although these may be considered as anticipations of MacArthur and Forrest's patent, the employment of zinc as used by them in the form of fine shavings was certainly novel, and has much to do with the early success of the cyanide process. Had zinc been introduced in the granular or powdery form, then commonly employed in laboratory work, the difficulties attending its practical application on a large scale are so great that in all probability the cyanide treatment of ores would not have met with the popular application so marked in its history.

In the year 1888 Werner von Siemens, of Berlin, applied for patents on the chief gold fields of the world for the extraction of gold and silver from ores, which consisted in treating ores with a cyanide solution, and afterwards precipitating the gold and silver from the separated solution by means of an electric current, using anodes of iron and cathodes of lead. The process was not applied on a working scale until 1893.

Thus it would appear that the extraction of gold and silver by the aid of cyanide began from mere suggestions and laboratory experiments, which conveyed to various minds the possibility of devising a process for dissolving out

\* *Die Chemisch-technischen Mittheilungen*, 1862-3, and *Otto's Chemistry* 1864.

precious metals from ores. One of the most obvious factors in favour of cyanide as a solvent, which has been appreciated by chemists from the first, is that the compound formed is of so stable a character that the mineral matter of the ore does not precipitate the dissolved metals, whereas when chlorine is used it is difficult to prevent the gold from re-precipitating. In other words, the gold and silver are more electro-positive in a cyanide solution than the pyrites or other mineral matter, while in a chlorine solution the reverse is the case.

Had Rae's process of 1867 been practically applied on a suitable ore and followed up by trained metallurgists, there is no doubt but that it would have developed in a few years into a successful cyanide process. As it was, and in spite of additional knowledge and subsequent improvements on Rae's ideas, it was not until the year 1891, when a plant was erected at the Robinson Gold Mining Company to treat tailings by the MacArthur-Forrest process, that the treatment of ores by cyanide had crystallised into a form acceptable to the mining world. From that time it may be said that the cyanide process, which has proved such a phenomenal success, dates.

The chief credit in proving the adaptability of cyanide to gold ores and in putting before the world a workable process is due to MacArthur and the Forrests, their colleagues, and the Cassel Gold Extracting Company of Glasgow; and not a little of the credit is due to that careful chemist, G. A. Darling, who successfully carried out the treatment with the first Robinson plant at a time when the available knowledge of practical difficulties was so meagre that success depended almost entirely on his technical skill. This will be the more appreciated if we quote an American authority, G. A. Stetefeldt,\* on new metallurgical processes:—"To invent a new process and perfect it technically is one thing—to make it financially a success is an entirely different affair. Inventors are only too often deficient in qualifications for managing their own business. If a new process is not a financial success at the beginning of its introduction, it may take years to obliterate the odium of failure. A process may be all right in theory, its technical points may have been worked out to perfection, but if it labours under the odium of many failures that may have nothing whatever to do with the process itself, it becomes difficult to instil into the minds of the public anything but failure. The successful introduction of new metallurgical processes seems to be a difficult matter, owing to the fact that general managers of mines are rarely men of professional education, and are incompetent to pass judgment on the merits of a new process. For this reason it is difficult to introduce a new process. On the other hand, if a mine does not pay, the manager is generally eager to try some new process that promises a great saving, but in most cases he cannot provide the necessary means for erecting a perfect plant."

"In conclusion, I will say lixiviation requires much more skill than amalgamation. So many variations of treatment being possible, much

\* *Eng. and Min. Jour.*, Oct. 4, 1890.

experimental work is needed to establish the most profitable method for a given ore."

Since the above was written, the successful introduction of the cyanide process has marked an era in the history of gold mining, not only by increasing the profits and diminishing risks to the investor, but by bringing into the industry a large number of professional men who take positions second to none in technology and finance. Stetefeldt's imputation no longer stands good in a general way, for we can now say, as to the general managers of gold mines, that they are for a large part as capable men as are to be found in any other industry.

## CHAPTER II.

### PRELIMINARY INVESTIGATIONS.

#### *Section I.*

**General Considerations.**—In determining the adaptability of the cyanide process to the treatment of gold and silver ores, the utmost accuracy in the mode of procedure is absolutely necessary if the results are to be of value. There are many important factors to be observed, any one of which may have a special bearing, not only on the construction of the treatment plant and the method of procedure in the process, but also on the method to be adopted for crushing and dressing the ore.

Amongst the various points to be considered are the physical and chemical states in which the gold and silver exist, as far as determinable; the method of crushing, such as dry or wet crushing, and the most suitable plant for this purpose—stamps, rolls, ball mills, etc., size of screens and sizing of the crushed ore; whether it is advantageous to concentrate, the adaptability to percolation or to agitation and subsidence, with the general arrangement of the plant; whether roasting is beneficial, or whether fine grinding is more advantageous; the dissolving of the gold from ores, raw or roasted, and of different sizes, by means of different strengths of solution; the consumption of chemicals and the precipitation of the dissolved metals. Such considerations would be very incomplete without an estimate, for each method investigated, as to cost of treatment on a working scale and probable profit. The profit is the most important of all points, for we are here dealing with a problem, not merely from a scientific point of view, but from a commercial aspect. To estimate probable profit with any degree of accuracy from preliminary investigations requires much judgment and long practical experience in the working of the process.

The several points above mentioned may be subdivided into others; and if each investigation is carefully made and systematically tabulated, it will be found that the trouble and expense it entails will be more than compensated for, disappointment will be averted, and the success of the selected process practically assured. The importance of such an investigation before proceeding to design a plant presents itself very forcibly when economic considerations are taken into account, for in treating a gold or silver ore there is often but a narrow margin between profit, in one case, and loss in the other.

Investigations of this sort should be entrusted only to one who has a

thorough theoretical and practical knowledge of the process, skill in laboratory manipulations, and some business ability—in reality, a man who has had a mechanical and chemical training, and practical experience in the complete treatment of gold and silver ores by wet processes.

No hard and fast rules can be laid down as to the method of carrying out an investigation and in treating an ore, but a few general outlines are here given as a guide, and these may be modified to suit any variety of circumstances.

**Sampling.**—The first important matter is to get a portion of the ore to be investigated. This should be taken so as to represent a fair average sample of as many thousand tons as it is practicable to procure. The sample cannot be too carefully taken, for this is the foundation on which the entire investigation rests.

The ore to be sampled may be taken from—

- (a) The mine.
- (b) An ore heap before crushing.
- (c) The current product as it leaves the mill in a crushed condition.
- (d) A heap of material already crushed.
- (e) Truck loads as they go to or from the works, or belt conveyors.

(a) The sampling of the ore in the mine is usually done by breaking pieces off the whole face of each stope at regular intervals. This does not appear to be a difficult task, but considerable judgment and intelligence are required in avoiding an undue quantity of abnormally rich veins or pockets or of more or less barren rock, in order to get a fair sample. Mine sampling is a business in itself, and should be entrusted only to those who are specially qualified for the work and disinterested in the results, but it is well that the metallurgist should know what to do with the sample when it is taken to the surface.

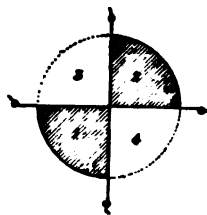


FIG. 1.—Sampling.

The sample ought to weigh at least 10 cwts., and, if the mine is extensive, may be as many tons. On the surface it is spread out on a hard clean floor, broken into pieces to pass through a 2-inch or 3-inch ring, and then shovelled into a conical heap. It is next thrown back into another heap, when the ore should have been properly mixed, and this operation may be again repeated, if thought necessary by the sampler. It is now levelled off into a flat circular-shaped heap of about 6 inches deep or more, and two strings are stretched across its diameter at right angles as in fig. 1, two opposite quarters are retained as 1 and 2, while 3 and 4 are thrown out. The part of the ore which is retained is again turned over and thoroughly mixed, and a similar circle is made and again quartered down until the sample is finally reduced to 1000–3000 lbs. This is then broken by a small hand or power stone-breaker to hazel-nut size, and is again reduced in bulk by quartering, or better by means of a Clarkson divider (page 10), to, say, 200 or 400 lbs. The latter is crushed to any desired degree of fineness by means of a 'dolly' or

hand stamp with water. The whole of the water and slime should be saved. If crushed dry, a small ball mill of the closed type answers the purpose, but care should be taken that as little as possible of the dust escapes. Every piece of ore belonging to the sample should be completely crushed to pass a sieve of the desired mesh and nothing left.

(b) When an ore heap is to be sampled it is usual to make a cutting right through its centre of a narrow but convenient width to work in; and as the ore is removed, it is dumped into a conical-shaped heap and then dealt with as already described under (a). When this heap is large it is of course reduced in bulk by quartering to a convenient size of 2 or 3 tons before breaking down to 2 or 3 inch gauge.

(c) Crushed ore as it leaves the mill may be either wet or dry, the former being most usual with gold ores. When crushed dry, samples may be conveniently taken by an automatic sampler as the ore is discharged from the mill. Such a sampler, to give satisfactory results, is not difficult to construct.



FIG. 2.—Sampling and Dividing Machine.

When, however, the ore is crushed wet, sampling becomes a more difficult problem. The ore as it leaves the battery tables is usually carried away by a launder, and at any point where a fall may be conveniently obtained samples are taken. This is done by taking a measured quantity of water and sand at regular intervals of every hour or so, by placing a vessel under the launder so as to take the whole of the pulp then passing, and next discharging the pulp caught into a large tank. This operation may be conducted by hand, but it is easy to devise an automatic appliance for the purpose which answers fairly well. The contents of the large tank are allowed to settle and the clear supernatant liquid is syphoned off at intervals, while the sand and muddy water are run off into a drying pan, where the superfluous water is evaporated and the ore thoroughly mixed.

This may be done once a day, or, where the ore is more uniform, once every two or three days.

In this way a very accurate average may be obtained, but the method requires cumbersome plant, is troublesome to deal with, especially in large mills, and involves much labour after the sample is collected.

In order to obviate these difficulties, a large number of automatic appliances have been devised, whereby small samples are taken continuously or intermittently from a narrow width of the launder, so as to reduce the bulk handled and save labour. But tailings flowing with water have such a tendency to concentrate that we have not, as yet, found an entirely satisfactory automatic sampler of this kind from which the results tally with those of the first method.

The sample should be dried at a temperature not exceeding 150° F.

until it contains about 5 or 6 per cent. moisture, in which condition it is reduced in bulk to about 200 to 400 lbs. by quartering and dividing as described under (a), or better by employing an automatic apparatus such as a Clarkson Divider. This apparatus is shown in fig. 2. It mixes the sample thoroughly, and gives a uniform product of any desired bulk. If the time of collecting the sample be noted, and the ore after drying be weighed, we can obtain a fair approximation of the quantity of material being daily produced. ✓

(d) It sometimes happens that the ore to be sampled consists of a heap of tailings which has accumulated from time to time. In this case we proceed by laying off two sets of equidistant parallel planes 8 to 60 ft. apart, at right angles to each other, and thus divide the whole of the heap into squares of equal area, measured horizontally, as shown in figs. 3. Pegs are inserted at the corners of each square, and a sampling rod is pushed down in these spots through the material to the surface of the ground beneath it. The whole of the cores drawn from the four holes of each alternate square are well mixed, reduced in bulk if necessary, then put into clean linen bags, numbered, sealed, and sent to the assayer, each set of holes being completed before proceeding with the next.

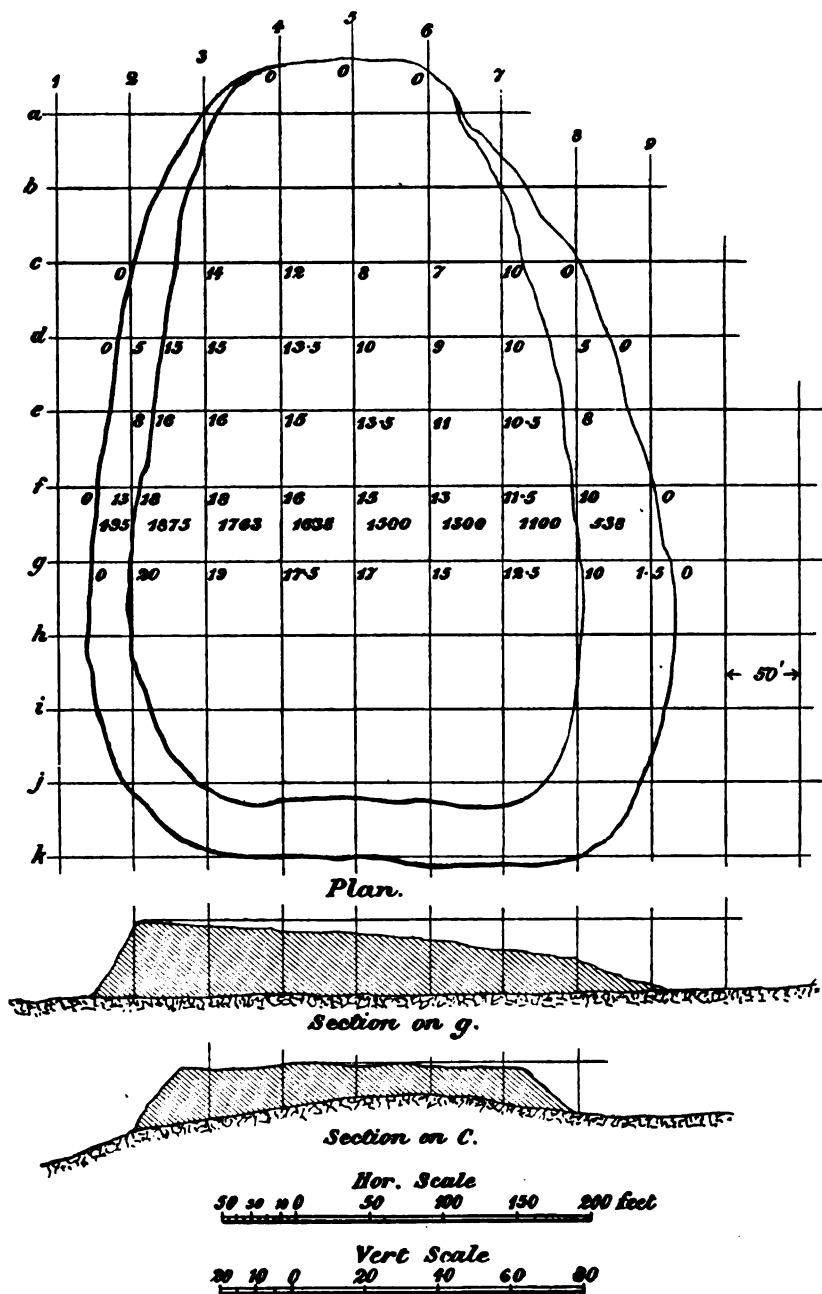
In this way a fair average of the assay value of the whole heap may be determined, but before this can be done with any accuracy it is necessary to know the cubic contents of the material in each square.

Referring again to fig. 3, the levels of the surface of the heap and of the ground on each side are taken with a level and staff in the ordinary way, and sections plotted showing the actual outline of the tailings and the real shape of the ground along any line of borings. The surface of the ground underneath the heap can often be sketched in with sufficient accuracy when the general surface of the country shows the inclinations to be constant, as in fig. 3a. If convexity or concavity exist, as in 3b, this must be determined by borings. The depth of material at the corners of each square, whether determined by direct borings or scaled off the plotted sections, should be marked on the plan at the corner of its respective square, as shown in part on fig. 3, and the contents below each square will evidently be the average of the depths at the four corners multiplied by the area, or, what is the same thing, the sum of the four depths multiplied by a quarter of the area. Thus, if the sides of the square were 20 feet, its contents in feet would be the sum of 4 depths in feet  $\times 20 \times 20$   
4 or sum of 4 depths  $\times 100$ , and this result

divided by the cubic feet in a ton of tailings gives the contents of each square in tons, which should be set down on the plan as shown between lines *f* and *g*. The quantity of gold and silver in the whole heap may then be calculated, and divided by the total number of tons, gives the average value per ton.

It sometimes happens that the heap is too deep for the sampling rod to reach the bottom, in which case several pits are dug at equal intervals until





FIGS. 3, 3a, 3b.—Measurement of Heaps.

the rod can be driven to the floor. The core drawn from below these pits may be mixed to form separate samples, and assayed and tested separately.

Heaps of concentrates are measured and tested in the same way, but the borings are made much closer, 4 to 8 feet apart being sufficient.

Very often a tank of ore is sampled in the same way, that is, by driving the sampling rod down to the filter cloth at equidistant points, the cores from which are thoroughly mixed and sent to the assayer, and the same method is again repeated on the residues after draining.

This method of taking samples from a tank is objected to by some men on account of channels being formed for the solution to flow down instead of flowing uniformly through the whole mass. This objection does not always hold, but there are certainly cases where the method has proved prejudicial, particularly when the ore has not been sized, or contains much slime with coarse sand.

(e) When the ore is conveyed to the tanks by the truck or car load, or when it is carried by a belt conveyor, an excellent opportunity is given to take a fair average sample. In the case of truck loads a short sampling rod, about 1 inch diameter and 2 feet long, is driven into each load as delivered, and the cores are collected together. The same method is employed with the residues when discharging into trucks. This method, however, entails considerable expense in labour. When a belt conveyor is used, samples may be taken at regular intervals by filling a small dish and discharging into a large vessel. This may be done automatically.

Discrepancies occur, whatever method of sampling is employed. These are to a certain extent unavoidable, owing to the great number of factors that are involved, many of which may vary unobserved. It is only on rare occasions that the estimated gold and silver in a quantity of ore corresponds with that accounted for after treatment; and when it does so, it is just as likely as not to be a coincidence rather than a sign of careful work. We can, however, nearly always rely on getting a fair approximation when the operations are conducted with care by qualified men, and more particularly where the ore has been previously sized. ✓

**The Sampling Rod.**—This is usually made from a length of ordinary gas-piping, about  $1\frac{1}{2}$  inches diameter, by cutting a slot lengthwise, taking out rather over one-third of the circumference to within a foot of the top end. A cross bar of about 2 feet long is then firmly secured to the uncut end, for the purpose of rotating and lifting the rod. The rod should be driven down into the sand without rotating, and when down to the desired depth is rotated several times and the core drawn up.

Another shaped sampling rod is similar to a screw auger. This is driven down into the sand with greater ease than the former, as it may be rotated in the direction of the screw, and the core is drawn up without rotating. Both rods bring up cores containing a larger proportion of the ore near the surface than near the bottom. This is a disadvantage which is more marked in the former than in the latter shape.

**The Testing Laboratory and Fittings.**—The testing laboratory is preferably attached to the assay laboratory, and should in fact be under the same roof, but form separate rooms, and consist of a testing room and a sampling room. In the testing room ought to be available apparatus for one or more small leaching and precipitating plants, capable of dealing with a few pounds to half a ton at a time, a good supply of measuring glasses, burettes, flasks and beakers, well made scales and weights, a set of bottles of ordinary chemical reagents for qualitative and quantitative analysis, and the use of assay and chemical balances. Facilities should be given for obtaining devices that might assist in rapid and accurate results. It is without doubt false economy to under-man or under-equip the testing and assay laboratories, as even such a small increase in extraction as 2 or 3 grains per ton, which is likely to result from good management, is sufficient often to cover all the laboratory expenses.

The sampling room should be well lighted and ventilated, and kept as free as possible from dust and draughts. Amongst the chief articles required are a sampling plate for moist ore made of  $6' \times 3' \times \frac{1}{4}"$  steel, well planished and kept as free as possible from rust and a hot plate of about the same size for dry samples. This is made, as shown in elevation fig. 4, of two plates  $\frac{1}{4}"$  thick, riveted to a rim of 3" channel iron and well caulked. The plate should be kept at a fairly uniform temperature, not exceeding  $150^{\circ}$  F. This may be conveniently done by heating the water in a boiler and circulating it

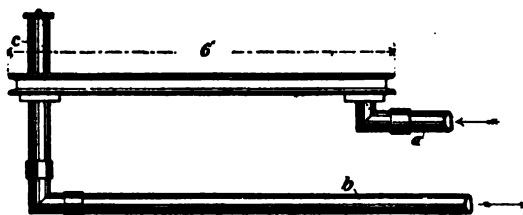


FIG. 4.—Hot Sampling Plate.

by means of tubes—(a) the hot pipe and (b) the cold—in the usual way; (c) is the water feeder. The plates are placed at a convenient height to work at standing, about  $2' 6"$  to  $2' 9"$ . Several scoops, trowels, and large spatules ought to be kept. A good supply of shelves and cupboards will be found very serviceable.

**Small Leaching Plant.**—The test of first importance is to determine the amount of gold and silver capable of being dissolved out of the ore in a given time, and the amount of cyanide consumed, for on these factors depends largely the adaptability of the process.

A simple, conveniently arranged plant, such as is shown in fig. 5, should form part of the apparatus of the testing room. A is a small stoneware jar, glazed, with a tubulure at the bottom, such as is sold by dealers in chemical apparatus, about 7" diam. and 10" deep inside measurements. A false bottom

is made of thick iron-wire sieving, about 36 holes to the square inch, supported at a height of  $1\frac{1}{2}$ " by iron-wire legs. This is covered with a piece of cheese cloth, and on that a piece of fine muslin or a piece of thin filter paper is placed. Both are well wetted and packed in round the wire gauze, and then the ore is filled in, either dry or moist.

A short bent glass tube is secured to the jar by a rubber stopper, and a long glass tube is connected with this one by means of a short rubber tube, to conduct the solution as it filters to its respective sump jar. These sump jars are made of glazed stoneware or glass, are four in number to each leaching jar—alkali, strong solutions, weak solutions, and water wash—and are readily obtainable at small cost. The solutions are preferably applied to the ore from measuring glasses, in order that the exact quantity may always be noted, and when drawn off again, measured and the quantity noted. Several such plants may be kept in operation at once. One or more larger leaching jars, 12 inches diameter and 24 inches deep, will also be found useful for testing poor tailings and residues.

**Precipitating Apparatus.**—It is advisable to keep a small zinc box for use when required, and this should be made so that the compartments are capable of being increased or decreased in area. The velocity of the solution passing through the box may then be varied within wide limits, and be made independent of the rate of the inflow.

An electrical precipitation box should also be kept, and also arranged that the velocity of the solution may be varied. For use with this apparatus a number of accumulators, ten or more, may be conveniently employed, and a delicate volt-meter and ampere-meter are essential.

Zinc precipitation tests on a small scale are usually not as satisfactory as on the working scale, but many important points present themselves in the laboratory which are not apparent on a large scale, and *vice versa*. On the other hand, precipitation by electrical methods gives, if anything, better results on a small scale than in a box of working size.

**Agitation Apparatus.**—It is also advisable to keep an agitation apparatus

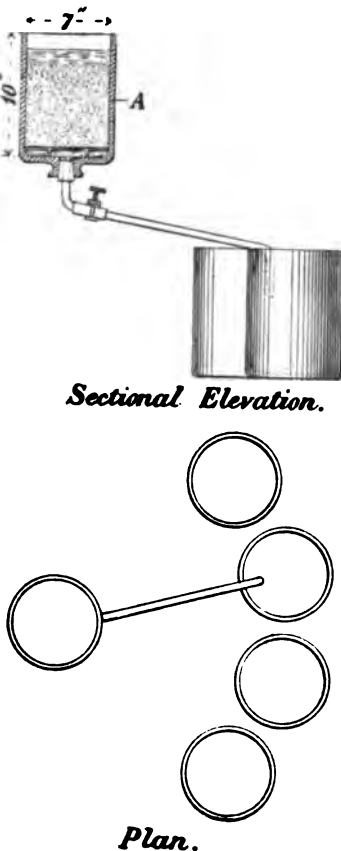


FIG. 5.—Small Ore Leaching Plant for Testing Samples.

in the laboratory, as ores are sometimes met with which cannot be successfully treated by the ordinary method of percolation, but are readily dealt with by agitation. Slimes may also be tested in the same plant.

Fig. 6 represents such a test agitation plant. *A* is an ordinary earthenware filter jar, which contains an agitator *B*, made of a disc of iron or lead with a hole in its centre. It is secured to a spindle *C*, carried by two brackets *E* and *F*, and these brackets are secured by set screws to a stiff vertical pillar *G*, and may be raised or lowered. On the top of the spindle is keyed a pulley *H*, driven from a vertical shaft *S* by a pulley *D*. *D* is secured by a feather and collars, and is adjustable to any height of *H*. The shaft is conveniently driven by a small electric motor or oil engine.

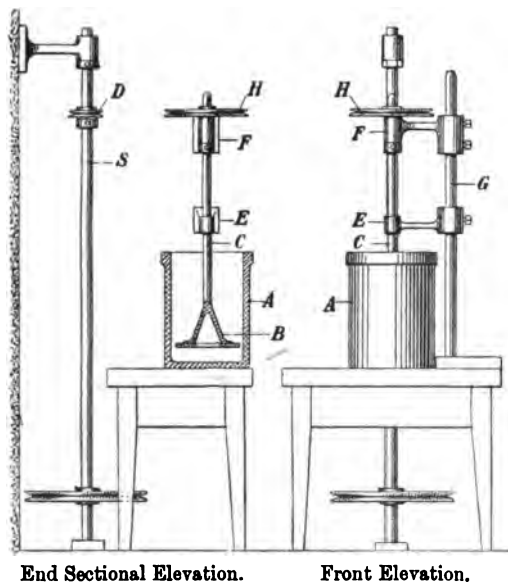


FIG. 6.—Apparatus for Agitation Test.

**Roasting Samples.**—This is best accomplished in a large sized muffle in a shallow clay dish, preferably shaped to fit the bottom of the muffle, and having sides 1 inch deep. The roasting is conducted at a low red heat, which is gradually increased to a dull red near the end of the operation, and the ore requires constant rabbling, especially at first. On no account should the ore be heated sufficiently to melt the gold or to form particles of matte, for in the first mentioned case the gold takes a spherical form and becomes extremely slow in dissolving in the cyanide solution, and in the second case the gold is hardly dissolved at all. A piece of asbestos packing or some clean sand placed under the roasting dish will be found of service in regulating the bottom heat.

The ore is weighed before roasting, and care must be taken that none is spilt during the operation. If weighed again after roasting, we get the loss

due to the presence of sulphides, arsenides, tellurides, and other volatile compounds. This should be noted and tabulated in the results of an investigation, as the quantity of these substances present may have some bearing on the results of extraction.

The cost of roasting can only be estimated from operations on a large scale, and will naturally vary in different localities and with the style of furnace used.

**Size of Sample.**—There is a tendency amongst chemists and metallurgists in testing samples of ore by cyanide to deal with too small a quantity, and thus magnify the range of experimental error. Thus it is not uncommon to find men who have been entrusted with investigations of importance dealing with 1 lb. of ore going 5 dwts. of gold to the ton; that is to say, such a minute quantity as 0.0625 gr. of gold. Even the most careful manipulator is sure to lose something, but that loss need not be much greater for a 10 lb. than for a 1 lb. sample.

A quantity of ore, which has been thoroughly mixed, should be taken sufficient for all necessary assays, cyanide tests, and re-trials, say 100 to 400 lbs. It is better to err on the right side, and keep abundance of the sample on hand in case re-trials become necessary. The least amount of ore allowed for assays should be enough for four assays of originals and four of residues, of say 4 A.T. each, for each test. A convenient quantity for the cyanide test is  $\frac{1}{100}$  of a ton. When an ore is sized in the laboratory into three or more sizes it may be too long and tedious an operation to get such a quantity of some one size, in which case a smaller proportionate quantity of that size would be taken to save time. A sufficient quantity ought, when possible, to be kept on hand to meet contingencies.

## CHAPTER III.

### PRELIMINARY INVESTIGATIONS.

#### *Section II.*

THE following are some of the more important tests to which an ore is subjected to determine whether or not it is suitable for cyanide treatment, and to ascertain the best methods of working and probable cost. The tests are divided into two classes, mechanical and chemical.

**Mechanical Tests.**—It is not the intention in these tests to deal with the subject of crushing the ore,\* although much may be learned from the investigations as to which method of crushing will be the best to adopt ultimately.

The prepared sample may be delivered in the sampling room as rock, crushed ore, either wet or dry, or as tailings.

The first operation is to examine the ore under a low magnifying microscope, and determine as far as practicable the physical state of the gold and silver and the associated minerals. Much may be learned by an experienced man and time often saved from such an examination. If the gold appears to have a smooth homogeneous surface, or appears as well defined crystals, it may be taken that the cyanide will, in all probability, dissolve only a small portion of it, and amalgamation will have to be resorted to if these gold particles are to be extracted. If, on the other hand, the gold is spongy or honeycombed in appearance, finely divided or invisible, it is possible it may be treated direct by cyanide. Then, again, the matrix may be of such a nature that crushing produces an excessive quantity of slime, as, for instance, when an ore is clayey or consists of certain schistose rocks; in which case wet crushing might produce, for the most part, a product impervious to the solution, while with dry crushing, leaching may become practicable with a larger percentage of the ore, or the ore may be quartz, and then either dry or wet crushing could be adopted. The decision would be influenced by the local conditions and the physical state in which the gold exists. When silver is present in a compound state, the microscope will often show which compound is acted on by the cyanide solution and which is not, and this information may lead to important results in treating silver ores.

The presence of metallic mineral matter, such as sulphides of iron, copper, antimony, zinc, etc., arsenides and tellurides, should always be determined by the microscope, their systems of crystallisation and the minerals identified, as far as practicable, in every unknown ore. This examination may lead to important results bearing on the method of procedure in treating an ore, and

save much time in experimenting. Also in cases of failure it may indicate the cause, and methods of overcoming difficulties may be thereby suggested. The microscope will also sometimes assist in determining whether or not the coarse particles of gold are amalgamable before or after cyanide treatment.

**Amalgamation Test.**—When an ore is to go through the amalgamation process it is usually crushed wet and amalgamated at the same time, leaving tailings to be treated by cyanide. But there are cases where an ore may be advantageously amalgamated after cyaniding, as, for instance, when much of the coarse gold is coated with sulphide films, as is often the case when the gold is found associated with stibnite and other sulphides. Such gold has a deeper colour than usual and amalgamates only with great difficulty, but it often happens that these films are easily removed by cyanide, thus leaving the gold in a clean amalgamable state. Amalgamation after cyaniding necessitates greater expense than when the operation is conducted with the crushing, but it has the advantage that dry crushing can be adopted if desirable. Then, again, there are many cases where the gold exists in such a state that amalgamation either before or after crushing becomes unnecessary.

It is usual in testing the sample to amalgamate before sizing, but the reverse order may sometimes be advantageously employed. The test in its simplest form consists in taking the weighed quantity of ore to be employed in the cyanide test and panning it with water in a prospecting pan, with an ounce or two of pure mercury. About 1 lb. of ore at a time is placed in the pan and panned in the usual way until the desired quantity is finished, and the whole of the tailings and water is washed into a vessel, preferably of enamelled iron or glazed earthenware. The mercury or amalgam is next carefully retorted, and this is conveniently done by putting it into a porcelain or smooth gold-annealing crucible, placed in a miniature retort over a regulated flame or in a muffle furnace.

The resulting bead is melted with or without fluxes, and if necessary parted, then weighed and the results tabulated. The whole of the tailings and water is saved for the cyanide test. When it is not required to know the quantity of free gold present, much time is saved by simply streaming the ore over a well 'set' amalgamated plate before proceeding with other tests. We have then a product similar to that of ordinary tailings from the amalgamation process. F. Hille \* suggests the use of a small ball mill as a convenient apparatus for crushing in making amalgamation tests.

**Sizing the Sample.**—To make a cyanide test anything like complete, it is advisable to classify the ore according to the size of the particles, just as would be done on a full working scale. But it would be difficult, if not impossible, to get a sized product on a small scale identical with that obtained on a working scale. However, an approximately similar classification may be got, either in the dry or wet way, which answers most purposes required for the investigation.

**Sizing Dry.**—This is done by passing a weighed portion of the material,

\* *Eng. and Min. Jour.*, p. 136, 1896.



after it has been dried at a temperature of 150° F., through a series of screens, say, for instance, 30-, 60- and 90-mesh. A covered set of ordinary assayers' sieves, 9" to 12" diameter, answers the purpose, and by this method four sizes are obtainable in one operation. The quantity retained by each of the sieves and that which passed all the sieves is weighed separately and the percentages noted. A portion of each size is taken for the assays, and other portions are weighed off for the cyanide tests.

The sieves require careful cleaning after use, and this may be conveniently done by gently tapping the screens with a short stiff bristle brush. The first pound or two of the sample first sieved might generally be discarded, and the sieves again cleaned before proceeding with the remainder. A separate set of sieves should be kept for rich ores and concentrates.

Table I. gives actual figures of a dry sizing test, and represents a convenient method of tabulating the results.

TABLE I.

Group No.	Held by 30-Mesh.						Held by 60-Mesh.						Held by 90-Mesh.						Passed 90-Mesh.					
	Assay of Originals.			Assay Value.			Assay of Total Gold.			Assay Value.			Assay of Total Gold.			Assay Value.			Assay of Total Gold.			Assay Value.		
	dwts. gra.			dwts. gra.			dwts. gra.			dwts. gra.			dwts. gra.			dwts. gra.			dwts. gra.			dwts. gra.		
	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.	Per cent. of Total Weight.
a	7	23	18	5	9	12.2	32	7	13	30.3	14.2	7	19	13.9	35.8	11	7	50.8						
b	5	8	19.2	6	9	23.0	32.6	2	14	15.8	13.4	6	4	15.5	34.8	6	9	41.6						
c	12	19	22.8	6	12	11.6	30	7	19	18.3	13.2	10	7	10.6	34	17	6	45.9						
d	14	3	16	9	13	10.8	31.6	9	18	21.8	14	11	23	11.9	38.4	18	5	49.5						

**Remarks.**—The ore is of quartzose nature, containing gold in a fine state of division. It was crushed dry in a ball mill and passed through a 20-mesh screen. The ore had not passed the amalgamation process.

It may be noticed that the sums of the gold percentages are not exactly correct, there being in one case more and in the other cases less than one hundred. Some discrepancy of this kind may be expected, especially in dealing with an ore containing free gold. It is not advisable to adjust figures in such a case, but leave them for comparison with the results by the cyanide test.

As a method of sizing an ore for leaching purposes, dry sieving does not give very satisfactory results, for an ore that just passes a 90- or 100-mesh screen gives a good leachable product, and for this reason only a rough idea of the quantity of slime present is obtained. The results, however, bring out the points that although crushed through a 20-mesh screen, about half the material in each case passed a 60-mesh screen, and that the material retained by the 30-mesh screen contained only 10 to 20 per cent. of the gold. It may be fairly assumed, pending further tests, that the ore has been crushed in each case sufficiently fine to liberate the gold.

Dry sizing with an air blast gives far more useful results than with sieves or screens, as the fine dust that forms slime may be more perfectly separated from the leachable products.

Table II. gives a case where dry sizing was done with an air blast on a large scale, and the results are tabulated in a similar manner to Table I.

TABLE II.

Sample No.	Assay of Originals.			1st Size.			2nd Size.			3rd Size.			4th Size.			Dust.		
				Per cent. of Total Weight.		Assay Value.	Per cent. of Total Weight.		Assay Value.	Per cent. of Total Weight.		Assay Value.	Per cent. of Total Weight.		Assay Value.	Per cent. of Total Weight.		Assay Value.
	dwts. gra.	Per cent. of Total Weight.	dwts. gra.	Per cent. of Total Gold.	Per cent. of Total Weight.		dwts. gra.	Per cent. of Total Gold.	Per cent. of Total Weight.	dwts. gra.	Per cent. of Total Gold.	Per cent. of Total Weight.	dwts. gra.	Per cent. of Total Gold.	Per cent. of Total Weight.	dwts. gra.	Per cent. of Total Gold.	Per cent. of Total Weight.
1	14	3	12	16	3	13.8	20	13	9	26	38.2	14	5	38.4	14	11	3	13.6
2	7	11	5.5	12	11	9.2	22	9	13	23.2	48.8	6	13	43	18.3	6	7	16.6
3	5	22	9	6	16	10.3	19.5	5	3	18.6	34.2	8	0	46.2	22.3	4	0	15.2
																15	3	10
																		9.7

**Remarks.**—Sample 1 is from schistose rock containing much mispickel. The gold was not discernible by the pan or by the microscope. None would amalgamate by the ordinary method of plates. Was crushed dry in a ball mill. Sample 2 is from quartzose rock from which 50 per cent. of the gold had been recovered by amalgamating over plates. Iron pyrites was present to the extent of about 2 per cent. Sample 3 is from quartzose rock containing clayey matter, iron oxide, and some pyrites. About 4 dwts. of gold per ton had been removed by amalgamating over plates. Both 2 and 3 were crushed wet with a stamp mill, dried and pulverised before sizing. Screens contained about 1000 holes to the square inch.

The first three sizes were good leachable products, the fourth leached fairly well, but left behind much moisture after draining, and the dust would not leach at all. The amount of dust shown in the table was that contained in the fifth chamber together with that lost from all sources. The figures were obtained by difference from originals.

It had been found that Nos. 1 and 3 could not be profitably treated by leaching before sizing, but by removing about 15 per cent. of the finer particles the remaining 85 per cent. gave a good leaching product. In the first case 13.6 per cent. of the gold remained in the dust and would probably pay to recover, while in the third case only 9.7 per cent. of the gold from a poor original would probably be unprofitable to extract.

**Sizing Wet.**—In this case the ore is weighed undried, and the moisture is determined from a small sample and allowed for.

The process of wet sizing is best carried out on samples in some form of elutriation apparatus, but it may be performed in a less perfect manner by means of a prospecting pan. The panning is done in the following manner:—A couple of handfuls of ore are placed in the pan at one time, and with a small

quantity of water is formed into puddle. The pan is then dipped into a large vessel and covered with clean water, is given a few quick semi-rotary movements, and at the same time gently lifted and tilted so that the light particles in suspension in the water flow over the edge of the pan. This operation is repeated several times until the light fine particles have escaped into the larger vessel, while the heavier and coarser particles are retained in the pan. This operation is again repeated on the ore which had escaped into the larger vessel, but in a more gentle manner, until the medium size particles are retained in the pan and the slimy matter has again passed into the larger vessel. The slime is then allowed to settle and the three sizes obtained. The method is slow and cumbersome, and no reliance can be placed on getting, even approximately, the same results in any two sets of operations.

A much more satisfactory result is obtained by employing a small elutriation apparatus, as shown in fig. 7. This apparatus consists of three inverted

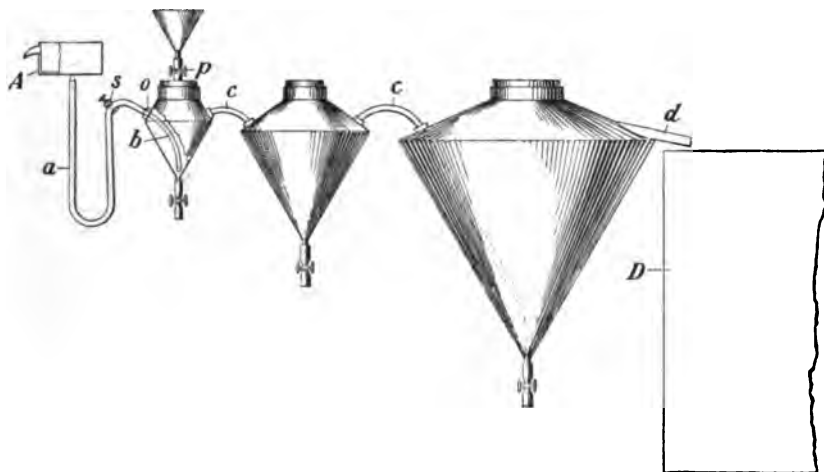


FIG. 7.—Elutriation Apparatus for Sizing and Classifying Samples.

hollow cones, having an angle of about  $33^{\circ}$  at their apices, and having diameters of bases in the proportion of 1 : 2 : 4. Other proportions are not of much importance. The cones are covered by hollow conical pieces, and fitted with wide mouths and rubber stoppers. In the stopper of the first cone is inserted a short glass tube, to which, by a rubber tube, is connected a funnel.

*A* is a clean water cistern, adjustable to different heights, and connected by means of a rubber tube *a* with a metal or glass tube *b*. This passes through a rubber stoppered tubulure *o* to the apex of the cone. A similar tube passes to the apex of each cone in the same manner, and a connecting tube *c* is inserted as shown to convey the water and particles of ore in suspension to the next cone.

The vessel *A* is adjusted to a suitable height and water is allowed to flow into it a little faster than it flows out through tube *a*, and is thus kept overflowing. When the apparatus is filled with water a clip *s* is closed and a

small quantity of ore in the form of sludge is fed from the funnel into the first cone by loosening the spring clip *p*. *p* is then closed and *s* opened, and the water is allowed to flow from the cistern *A* until it is practically clear as it leaves the discharge pipe *d*. At the apex of each cone is a short tube of rubber with a spring clip for the purpose of discharging the ore into separate vessels at intervals. The operation is repeated until a sufficient quantity of each size is collected. The fine and light particles flow into cistern *D*, where they are allowed to settle. The clear water having been syphoned off, the residue is next dealt with by again repeating the operation, but with a much reduced pressure of water at *A*. The pressure is in fact reduced to  $\frac{1}{18}$  of the original.

In this way seven sizes of an ore may be obtained; and if each size is submitted to a percolation test, we can readily find out the proportion of non-leachable product present, and in what sizes the gold and silver chiefly exists. Other factors may also be rendered apparent, as, for instance, whether or not the ore has been crushed sufficiently fine to liberate the gold and silver, and tests may be made on each size, by leaching with cyanide, to find out where the bulk of the metal remaining in residues is to be found.

Table III. shows results of four tests on the same samples as Table I., but sized wet.

TABLE III.

Group No.	Assay of Originals.			Coarse.						Medium.						Slime.					
				Per cent. of Total Weight.			Assay Value.			Per cent. of Total Gold.			Per cent. of Total Weight.			Assay Value.			Per cent. of Total Gold.		
	dwts. grs.			dwts. grs.			dwts. grs.			dwts. grs.			dwts. grs.			dwts. grs.			dwts. grs.		
<i>a</i>	9	18		9.5	20	4	20	64.9		8	19	58	25.6			9	3	24			
<i>b</i>	5	8		8.1	5	6	8	70.9		4	13	60	21.0			7	12	30			
<i>c</i>	12	19		6.8	18	4	10	69.2		9	23	54	24.0			16	12	31			
<i>d</i>	14	3		14.2	20	10	21	68.6		11	17	57	17.2			16	23	21			

**Remarks.**—These samples were sized in a prospecting pan. See remarks below Table I.

In comparing the results of Tables I. and III., it may be noticed that Table I. shows that over 34 per cent. in each case passed the 90-mesh sieve and a large amount of slime might be anticipated, but Table III. corrects that impression by showing a minimum of 17.2 per cent. of slime in group *d* and a maximum of 25.6 per cent. in group *a*. These quantities of slime would most probably be still further reduced with more perfect sizing apparatus, such as

described above, and thus the quantity of ore capable of being dealt with by percolation increased.

**Percolation Test.**—The next mechanical test is to determine what percentage of ore is capable of being treated in leaching tanks and what quantity would have to be dealt with by other ways. The cost of leaching in tanks is less than by other means, and it is therefore advisable to treat the largest quantity practicable by this method. This maximum quantity is determined by first sizing a sample and testing each size separately, when the limit will be found at which percolation in tanks must cease.

The rate of percolation is determined sometimes by the time the solution above the ore sinks to a certain depth; thus, over 3 ins. per hour is good,  $1\frac{1}{2}$  ins. per hour is fair, and  $\frac{3}{4}$  in. per hour bad. Less than  $\frac{3}{4}$  in. per hour is usually uneconomical. Louis Janin jr. says\* that tailings from the Bremen Mill, Silver City, New Mexico, ground fine in pan amalgamation, leached at the rate of  $\frac{1}{16}$  in. per hour, although aided by pump suction under the filter. He gives the rate for raw ores crushed dry by rolls through 20-mesh screen at 6 ins. per hour as a minimum.

At Aspen, Colorado, in a Russell plant, the natural leaching rate was 13 ins. per hour, reduced to 10 ins., however, after the ore had been washed.† The rate of percolation varies widely with ores of the same fineness, being dependent largely on the method of charging the tanks.

A better method than the above is to compare the porosity of the ore by comparing the quantity of moisture retained after draining. The greater the percentage of moisture retained, the smaller the porosity and the slower the percolation. In these tests the time allowed per foot of depth of tank should be that which could be economically given on a large scale.

In some classes of ore the amount of slime is so small and of such a character that, when mixed evenly with the leaching products, it does not materially interfere with percolation or with the extraction of the metal. In such a case it might be most economical not to size at all.

It is, however, generally advantageous to classify the ore into three sizes—coarse, medium, and slime; and where the amount of slime is large, a fourth size may materially increase the quantity of ore capable of being dealt with in the leaching tanks. Slime may be defined as that ore which cannot be economically treated in the leaching tanks on account of the fineness of the particles. The next size should be of such a degree of fineness that after draining it would hold not more than 40 per cent. of moisture.

To compare the porosity of the different sized products the apparatus shown in fig. 8 may be employed. *A* is a tube of 1 to 2 ins. diameter and 4 or 5 ft. long, connected at its lower end to a measuring glass *B* by means of rubber tubing *C*. The tube *A* is filled to a mark, say 4 ft. from the bottom, with the ore, previously dried, and the net weight noted. A rubber stopper *a* fitted with a short length of glass tube serves as a means of connecting *C*.

\* *Mineral Industry*, vol. i. p. 253.

† Wm. S. Morse Prescott, *Trans. Am. Inst. Min. Eng.*, xxv. p. 143.

The hole is covered over with a piece of muslin to support the sand. The glass *B* and tube *C* are next filled to zero with a clean cyanide solution of working strength, and the zero mark is brought on the level of *a*. The cock *T* is then opened and *B* is very gradually raised until the solution in *A* rises to a point about 3 ins. above the ore surface and the solution in *B* is on the same level. *B* and *C* are next lowered and allowed to lie on a sloping board *D*, which may be adjusted to any angle between horizontal and vertical according to the suction found desirable, and then maintained in the same position for any set of experiments. Time observations may then be made of the fall of the solution in *A*. Allow the ore to drain for as many minutes per foot of depth as would be employed on a large scale. Finally, disconnect *C* from *a* and weigh *A* to find the weight of moisture retained in the ore. The less the moisture retained the more suitable is the ore for treatment by percolation in tanks, leaving out, of course, the question of dissolution of the metal.

Table IV. shows the results of a sample of ore which had been sized so as to reduce the quantity of slime to a minimum.

TABLE IV.

No. of Size.	Per cent. of Total Weight of Ore Sample.	Weight of Ore Dry of Equal Bulks.	Weight of Ore after Draining.	Moisture retained per cent. of Ore.	Remarks.
1	31.4	1800	1881	4.5	Percolated freely.
2	18.2	1455	1556	7	
3	21.2	1605	1755	9.3	
4	12	1226	1367	11.5	
5	8.2	1005	1148	14.2	Percolated 1 in. in 1½ hours. Would not percolate in time allowed. Ore loss 0.5 per cent.
6	5	954	...	...	
7	3.5	902	...	...	

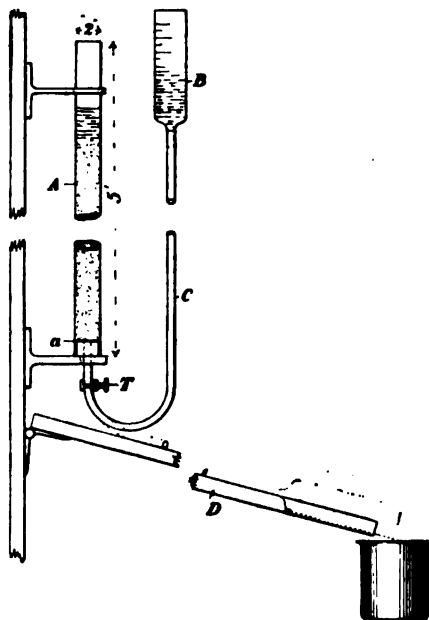


FIG. 8.—Apparatus for Testing the Porosity of Leaching Material.

This ore was crushed dry and consisted of schistose rock containing arsenical pyrites. It was found too impervious to treat direct in the ordinary leaching tanks, but by close sizing 91 per cent. could be dealt with in this way.

**Decantation Tests.**—These tests are very easily made by thoroughly agitating known weights of slime and dilute cyanide solution with different percentages of some coagulating substance, allowing the mixtures to stand, and comparing the depths or volumes of the sediments. Ordinary cylindrical measuring glasses of 1000 c.c. answer as the apparatus for these tests. Weigh off equal weights of slime, say five lots of 500 grms. each, and put each into a 1000 c.c. measuring glass together with different percentages of some coagulating substance, such as lime, say 0.1, 0.2, 0.4, 0.6, 0.8 per cent., add a dilute cyanide solution, say 0.05 per cent., a little at first, and thoroughly agitate; then fill up to the 1000 c.c. mark. Allow these to stand about three or four hours, taking care that the temperature remains constant, and note the depth of sediment in each case. The best percentage of lime is that which gives the greatest amount of clear solution for decantation. In some cases a maximum point is soon reached, whereas in others the most economical amount of lime has to be determined partly by cost. Beyond a certain amount of lime no advantage is derived, but often subsidence is retarded.

Caustic lime is generally employed in the cyanide process, as it protects the cyanide, but it is not as good a coagulating substance as other compounds of calcium. Thus calcium carbonate (chalk) is often more rapid in its action and produces a more dense sediment, which is not so easily disturbed while decanting as when the hydrate is used. (See *Slimes*.)

## CHAPTER IV.

### PRELIMINARY INVESTIGATIONS.

#### *Section III.*

**Chemical Tests.**—The question of chemical testing has been somewhat elaborately treated by many able writers, but under this heading only those tests are dealt with here which are likely to be of use in the ordinary routine work in the cyanide process.

Amongst the first chemical tests is the assay of ore, but this is now such a cut-and-dry subject that we need offer no apology for not reproducing what has been so exhaustively dealt with by many well known authorities.\*

It only remains for us to reiterate the opinions of W. Bettel, G. A. Darling, and others as to the importance of using a sufficiency of ore for the assay, say 6 to 12 A.T., when dealing with poor ores, tailings, and residues, and this may in some cases be increased or duplicated. When accurate investigations are being made, the pulverising of the ore to pass a sieve of about 90-mesh ought to be insisted on, and three or four assays made from each sample should give approximately the same result. For daily assays of originals and residues, assays in duplicate that correspond closely are usually found sufficient. The originals may be of 4 to 8 A.T. and the residues of 6 to 12 A.T. For determinations of gold and silver in solutions, about 30 to 200 A.T. may be taken according to richness. Each sample is evaporated in a large porcelain dish, the bottom of which is covered with litharge. The residue, when quite dry, is completely scraped out and fluxed, melted and cupelled. Crosse's method, when only gold is to be determined, is preferred by some chemists, as large samples may be dealt with and much time saved. This consists in acidulating and boiling, and then adding silver nitrate or copper sulphate in excess to the cyanide solution, which precipitates practically the whole of the gold. The solution is then filtered and the precipitate dried, scorified, and cupelled.

The object of using large quantities of material in assays is to get large beads for weighing, as a small bead magnifies the want of sensitiveness of the balance, whereas a larger one may increase the sensitiveness to a maximum. Thus 100 beads weighed separately on a fairly sensitive balance and added together showed an average ore of 19 grs. per ton, but when all the beads were weighed together it showed that the ore averaged 12.5 grs. per ton. It is therefore incumbent on the assayer to take large charges of ore, so that the

\* See *The Metallurgy of Gold*, by T. K. Rose. *Handbook on Gold Milling*, by Henry Louis.



beads may be as near as practicable of such weights as will give a maximum sensitiveness to the balance. It must be remembered also that there is always a loss in each stage of the assay—potting, cupelling, and parting—which is more or less unavoidable, and undoubtedly the percentage loss is greater with small beads than with larger ones. Another matter of importance is the size of the crucible. This should in all cases be sufficiently large to reduce to a minimum the risk of boiling over.

**Testing the Cyanide Solution.**—This is usually done by the well known method of adding to a measured quantity of the solution silver nitrate of a standard strength from a burette, drop by drop, until a whitish or opalescent turbidity just appears.

The standard  $\text{AgNO}_3$  is usually made by dissolving 13.04 grms. of the re-crystallised salt in 1 litre of distilled water for testing strong cyanide solutions, and half that strength for weak solutions.

Then for strong solutions every

$$0.1 \text{ c.c. } \text{AgNO}_3 \text{ added to } 10 \text{ c.c. KCy} = 0.01 \% \text{ KCy,}$$

or for weak solutions every

$$1 \text{ c.c. } \text{AgNO}_3 \text{ added to } 50 \text{ KCy} = 0.01 \% \text{ KCy.}$$

This method is simple and works well with pure KCy, but with commercial cyanide, which contains varying quantities of NaCy and other compounds besides KCy, the estimation is unreliable. Further, when a solution has been in use some time, and more particularly when zinc precipitation has been employed, it becomes of a very complex nature, and the end reaction of the titration is so indefinite that no reliance can be put on the results. To overcome this difficulty the test has been modified for impure solutions by the addition of a few drops of strong potassium iodide as an indicator, which gives more definite end reactions, and answers in a rough way for determining relatively the strength of the solution; that is, all the cyanides present which have any solvent action on gold.

J. E. Clennell,\* who has dealt with this subject exhaustively, points out that "the test should always be made in precisely the same manner." This is owing to the fact that the results are of a relative rather than of an absolute nature. The solution to be tested is measured in a graduated cylinder or in a pipette, say 50 c.c., and to this is added 5 c.c. of the indicator, made by dissolving 10 grms. KI and 40 grms. NaOH in 1 litre of water. "The liquid is titrated with standard silver nitrate (6.519 grms. per litre) until a distinct yellow coloration is obtained, disregarding any white turbidity. This latter may sometimes be removed by adding ammonia, which, in moderate amounts, does not affect the accuracy of the test—

$$1 \text{ c.c. of } \text{AgNO}_3 \text{ used} = .01 \% \text{ KCy (equiv. to 'total cyanide').}"$$

It must, however, be understood that there is no relation between the 'total cyanide' found present and the dissolving action of the solution on gold

\* *Trans. Inst. Min. and Met.*, May 1903.

and silver. For this reason two different solutions, containing, by the test, the same quantity of cyanide, may have very different dissolving effects.

Turbid solutions that are not cleared by simply filtering through paper are usually cleared by the addition of caustic lime. This treatment, however, more or less affects the results.

**Testing the Solution for Alkali.**—It is of great practical importance to have an excess of alkali in the working solutions. This is of service in assisting the dissolution of the gold and silver, and in protecting the cyanide from decomposing by 'acid' in the ore or carbonic acid in the air.

The alkali test as given by Clennell \* "is done (accurately in absence of zinc) by adding silver nitrate till a slight turbidity is produced, adding phenolphthalein to this turbid solution, and titrating without filtering with  $\frac{N}{10}$  acid." When zinc is present L. M. Green's † method is used, which consists in adding an excess of potassium ferrocyanide before titrating with the silver nitrate.

**Determination of Alkali Consumption.**—Gold and silver ores usually contain what is technically termed 'acid,' i.e. products of oxidation of sulphides, arsenides, etc., being a mixture of compounds soluble and insoluble in water, which have the properties of neutralising alkalies and rendering the cyanide unavailable for dissolving gold and silver. When an ore has been crushed and allowed to be acted on by moisture and the atmosphere, or when an ore is taken from a partially oxidised zone in the mine, it is often found to contain so much soluble 'acid' as to be readily indicated by litmus. In such a case it may or may not be economical to wash the ore with clean water before treatment with the cyanide solution. The acid is, however, usually neutralised by an alkali before adding the cyanide solution.

A suitable method of determining the quantity of alkali neutralised by the soluble acid is that given by H. Van F. Furman. ‡ "Agitate 10 grammes of the pulp for ten minutes with 50 c.c. of water; filter, and test the filtrate with litmus paper for acidity. Should acidity be shown, wash the ore until the washings no longer give an acid reaction when tested with litmus paper. Now titrate the total filtrate with decinormal caustic soda solution until the neutral point is observed, using litmus as an indicator."

$$\begin{aligned} 1 \text{ c.c. } \frac{N}{10} \text{ NaOH on 10 grms. ore} &= 0.88 \text{ lb. of NaOH per long ton} \\ &= 0.8 \text{ lb. } \quad \quad \quad \text{short } \end{aligned}$$

10 grms. of ore is rather a small quantity, and might advantageously be increased to 100 or 200 grms.

A more practical method is to employ a standard solution of such a strength that decimals of a pound per ton may be read off the burette without

\* *Trans. Inst. Min. and Met.*, May 1903.

† *Ibid.*, vol. x. p. 29 et seq.

‡ *Trans. Am. Inst. Min. Eng.*, 1896.

calculation. Such a method has been suggested by Feldtmann as follows:—  
Make a standard solution containing 1 per cent. of NaOH.

Weigh off 224 grms. of ore for a long ton.  
 „ 200 „ „ short „

Then to total filtrate add the standard solution

1 c.c. 1 per cent. NaOH solution = 0.1 lb. NaOH per ton.

In this way it may be determined whether or not the cost of washing the ore is going to be compensated for by the saving in alkali. It is generally a problematical question, for the ore is sure to consume a large quantity of alkali, even after the most careful washing.

When an ore has been washed, or when it is treated direct, the total alkali required may be determined (1) in terms of caustic soda, (2) in terms of lime, as follows:—

(1) Place  $\frac{1}{100}$  of a ton of the ore in a filter jar, fig. 5, page 15. Pass through the ore one-third of its weight of a 3 per cent. caustic soda solution, next add two-thirds its weight of water. This makes a solution of equal weight to the ore and averaging 1 per cent. Thoroughly agitate the filtrate and put 224 or 200 c.c. into a boiling flask and titrate with standard sulphuric acid, using methyl orange as an indicator.

Alkali in 1 per cent. solution = 22.4 lbs. per long ton.  
 „ „ „ = 20 lbs. „ short „

Make the standard sulphuric acid containing 12.25 grms. per 1000 c.c. (or 1.23 per cent.).

1 c.c.  $H_2SO_4$  = 0.1 lb. NaOH per ton of ore.  
 Let  $x$  = Number of c.c.  $H_2SO_4$  required to neutralise 224 c.c. NaOH.  
 $y$  = „ „ „ 200 „ „  
 Total alkali loss =  $22.4 - x(0.1)$  lbs. per long ton.  
 „ „ =  $20 - y(0.1)$  lbs. „ short „

The total alkali loss is that which combined with the 'acid' of the ore together with that which remained '*adsorbed*' after washing. The above test gives about what may be expected to be consumed on a working scale.

(2) It is more advantageous to employ lime instead of caustic soda as a neutralising agent, when obtainable at a reasonable price. The latter has much less power to coagulate colloidal precipitates, present in many ores, than lime; and although these precipitates may not interfere appreciably with the dissolution of the gold, they usually become troublesome in the precipitation boxes. Sometimes, also, when caustic soda is used, these colloids become unstable in the solution at different stages and retard percolation. On the other hand, lime has the property of precipitating these colloids in such a state as not to interfere with percolation.

The most suitable quantity of lime is determined in the following manner: first in a preliminary way, and afterwards by a more exact method.

Weigh off a number of equal samples of ore, say four of 200 to 300 grms., and add to each a different proportion of lime, say 0·1, 0·2, 0·4, 0·8 per cent. Mix thoroughly, place in flasks, then add to each sample equal measures of cyanide solution, say 200 c.c. of 0·2 per cent., and agitate, at intervals of ten minutes, for one hour, filter a portion of each solution, and test for cyanide (page 28). These results should show approximately the quantity of lime to use, for beyond a certain amount no saving in cyanide will be effected.

The amount of lime required may be more accurately determined as follows:—The preliminary test having shown where the saving in cyanide stopped, say, for instance, between 0·2 and 0·4 per cent., take three samples of the ore, each  $\frac{1}{160}$  of a ton, thoroughly mix with 0·25, 0·3, and 0·35 per cent. of lime, place in filter jars and run on 0·2 per cent. cyanide solution one-third weight of ore, without addition of alkali. Put the solution through a second time and test for cyanide. The results should indicate about the least quantity of lime necessary.

It must be remembered that when an ore is sized, each size should be tested separately, as the finer the ore the greater is the quantity of alkali and cyanide consumed. (See chapter on *Slimes*.)

When making tests on pyritic slimes, the moist material should be used without drying, because, as pointed out by W. A. Caldecott,\* the process of drying oxidises the reducing substances present, and thus effectually vitiates the test. The quantity of dry slime in the moist material can be calculated by the following equation† when the specific gravities of the dry slime and of the pulp are known.

Let  $s$  = specific gravity of dry slime.

$a$  = specific gravity of pulp found by weighing equal volumes of water and pulp.

$P$  = percentage by weight of dry slimes in pulp.

$$\text{Then } P = \frac{100 s(a-1)}{a(s-1)}.$$

Slimes are tested in a small agitating apparatus, fig. 6, page 16. This is done by agitating a weighed quantity of slime with about twice its weight of water, and adding at intervals of about ten minutes the alkali or lime in small percentages until neutral to litmus paper. At a slight excess beyond this point note the amount required, next add a known weight of solid cyanide or strong solution, agitate well, filter a portion of the pulp, test the filtrate, and note the loss. Other samples may be tested in the same way, with a greater or less amount of lime or alkali and the same quantity of cyanide, for comparison. There will always be a loss in cyanide due to other causes than acidity, which need not be gone into here. (See chapter on *Slimes*.)

Magnesia may be used in place of lime, but a larger quantity must be employed. It has, however, certain advantages over lime in some cases.

\* *Jour. Chem. and Met. Soc. of S. Africa*, July 17, 1897.

† W. A. Caldecott, *ibid.*, vol. i. No. 3.

**Oxidation of Ore Test.**—Much saving in alkali and cyanide may sometimes be obtained by oxidising the ferrous oxides to ferric oxides, by means of air or an oxidising agent, before applying the cyanide solution. An apparatus for testing samples by oxidising with air is shown in fig. 9. *A* is an ordinary filter jar containing the sample of ore ( $\frac{1}{100}$  tons). This jar

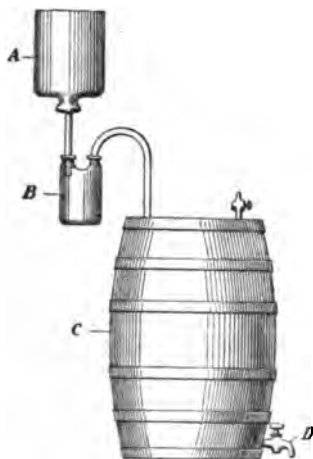


FIG. 9.—Apparatus for Testing the Oxidising Effect of Air on Samples.

is connected by a glass tube with one mouth of a Wolff bottle *B*, and the other mouth is connected by an iron tube with a cask *C*, of 40 to 60 gallons. The cask is placed at a level below the filter jar and is filled with water. The tap *D* is next opened and the water is allowed to escape at a known rate into a measuring vessel. A measurable volume of air is thus drawn through a known weight of ore in a known time. The ore is sprayed at intervals with a dilute alkaline solution of known quantity, which is caught in the Wolff bottle to be tested and the loss noted. A cyanide solution of 0.2 per cent. is then passed through the ore and the loss noted. This is compared with another sample of the same ore, treated direct with alkali and cyanide, and the costs in both cases estimated.

Another method of reducing the consumption of cyanide is to run a dilute solution of an oxidising agent through the ore before adding the cyanide solution.

Sodium peroxide is a suitable oxidising agent for this purpose. If the ore is first treated with a dilute alkaline solution and afterwards with a dilute solution of an oxidising agent, as potassium permanganate or ferricyanide, a saving in cyanide is effected. The most economical strength of such solutions can only be found by experiment on ores with different strengths of solution, results noted, and costs estimated.

Ore treated by agitation may have the consumption of cyanide greatly reduced by blowing air into the pulp at the centre of the tank near its bottom, and adding at intervals small quantities of lime or alkali before adding the cyanide. Oxidising agents may also be used instead of air by adding small quantities at intervals before the cyanide.

**Gold and Silver Dissolving Test.**—A test of great importance is to determine how much of the gold and silver in an ore is capable of being dissolved by the cyanide solution in a given time. This naturally varies with the form and character of the gold and silver, the mineral constituents of the ore, the size of the ore particles, and strength and impurities of the solution.

The test is best conducted in the same manner as on a large scale, adhering to the various factors in the working of the process as closely as

practicable. Size of plant undoubtedly has some influence on the dissolving of the metals, but this is not sufficient to interfere with the value of a test on a small scale. One often hears it asserted that the extraction on a large scale gives higher results than on a small scale, but this is doubtful if the conditions are similar. We have generally got better extractions on a small scale, and this is what may be expected, as the operations are more under control.

A small stoneware filter jar 10 in. deep and 7 in. diameter, and three or four similar sized jars as sumps, shown in fig. 5, page 15, answer very well for these tests. The filter jar is charged with ore ( $\frac{1}{100}$  ton), and the acid having been neutralised, 7 lbs. of strong solution of, say, 0.2 per cent. KCy is run on, and a portion is immediately run off until the solution and the ore are on the same level, the balance being allowed to remain in contact, say, twenty hours. This is next drained off slowly into its sump, allowing four hours. A weak solution, say 6 lbs. of 0.05 per cent., is next run on, a portion being drawn off immediately and run into the strong solution sump until equal in quantity to the strong solution originally applied. Contact is again allowed of twenty hours, and four hours for drawing off. A water wash of 7 lbs. is finally run on, and in an hour or two as much is drawn off into the weak solution sump as makes this solution equal in quantity to that originally applied. The water wash may then be allowed to run off immediately, but slowly, so as to take, say, twenty-four hours from the time of applying.

The total quantity of the three solutions used varies very much in practice, from  $\frac{1}{2}$  to  $1\frac{1}{2}$  times the weight of the ore, and the same proportion should be adhered to in small tests as would be used on a working scale. The time, also, of running off the solutions should be regulated so as to be approximately the same as on a large scale. The above is only one of a great many different combinations in the method of procedure, and these may be modified and varied to a great extent, often with advantage. It is well, however, in making any variation to vary only one factor at a time.

When each sump has its full complement of solution returned, it is tested for cyanide and the percentage noted. This is compared with the original solution, and the loss in cyanide determined in pounds per ton of ore treated. The ore residue is next assayed, and the sump solutions are evaporated to dryness and the metal extracted and weighed. The weight of bullion obtained multiplied by 100 gives the weight extracted per ton.

The ore residue may be again treated in the same manner, or with stronger or weaker solutions, to determine whether a larger percentage of the gold and silver is capable of being dissolved out. When this is the case the method of procedure may be varied until a maximum extraction is obtainable in a given time, with a given proportion of solution and with a minimum loss of cyanide.

A complete record of the method of procedure should be kept, and the results tabulated in some convenient form as on the next page.

TABLE V.

Experiment Number	Weight of Ore in lbs.	Assay Value.		Strong Solution.					Weak Solution.					
		Originals.	Residues.	Weight of Ore.	Per cent. KCy on.	Per cent. KCy off.	Hours' Contact.	Consump. per Ton Ore lbs.	Weight of Ore.	Per cent. KCy on.	Per cent. KCy off.	Hours' Contact.	Consump. per Ton Ore lbs.	
		dwt.	dwt.											

Experiment Number.	Weight of Ore in lbs.	Water Wash.					Consumption KCy Total per Ton Ore.	Gold extracted per Ton Ore.	Per cent. Extraction.
		Weight of Ore.	Per cent. KCy on.	Per cent. KCy off.	Hours' Contact.	Consump. per Ton Ore lbs.			

A large space should be left for remarks and comments, and all observations, if recorded concisely, may be found of great service in other experiments on the same ore, or when working on a full sized scale. It is highly desirable when practicable to make tests, also, with solutions that have been in use for some time, in treating similar ore to that being tested. But if zinc precipitation has been in use, a difficulty is experienced in testing the strength of the solution for making comparisons (see page 28).

A great many tests must be made on an ore to determine the most economical and efficient method of proceeding, and these might be very widely varied with regard to time of treatment, quantity and strength of the cyanide solution, and the mechanical texture of the ore.

In the practical working of the process it is also of importance to make frequent tests on samples of the residues, with a view to determining the quantity of dissolved gold and silver present, and the quantity capable of being dissolved by longer treatment, or by varying the strength of the solution.

If the ore has not been sized, it would be instructive to size a sample of the residues and determine in what size the gold or silver chiefly exists. The following is a case in point given by G. A. Goyder.\* A sample of cyanide residue containing 1 dwt. 13 grs. per ton of 2240 lbs. was treated

\* *Chemical News*, Aug. 23, 1895.

for forty-eight hours by continuous percolation and 8 grs. were extracted. This residue, containing 1 dwt. 5 grs., was dried and sized into three sizes by sieving as follows :—

Retained on 30-mesh = 2 per cent. assayed, 8 dwts. 3 gra. per 2240 lbs.

„ „ 60-mesh = 29 „ „ 2 „ 7 „ „

Passed 60-mesh = 69 „ „ 0 „ 14 „ „

From this it is calculated that the coarse particles in a ton of ore weighed 45 lbs. and contained 3 grs. of gold ; the medium, 650 lbs., contained 17 grs. ; and the fine, 1545 lbs., contained 11 grs.

Here we get a problematical question. It may be assumed that if the 2 per cent. of coarse particles were crushed to pass a 30-mesh, the residue would contain only 2 dwts. 7 grs. per ton, while if crushed to pass a 60-mesh the residue would contain only 14 grs. per ton. If the ore before cyaniding had been sized into three sizes as above, we should have had only 45 lbs. to crush in every ton treated to pass a 30-mesh for 5 dwts. 20 grs. per ton, and if crushed to pass a 60-mesh, for 6 dwts. 13 grs. per ton. Again, with the medium size we should have to crush 650 lbs. in every ton treated to pass a 60-mesh for 1 dwt. 17 grs. It is probable it would pay to re-crush the 45 lbs. per ton, and under favourable conditions it might even pay to re-crush the 650 lbs. per ton and re-size. Much will depend upon the increase in the production of slime, but in a well designed sizing plant the quantity of unleachable slime formed need generally be very small. There are, of course, exceptional cases, such as when the ore consists largely of schistose rock, when the quantity of slime produced is excessive.

In another case a sample of original ore was carefully sized in the wet way into four sizes, and each size was tested by itself, as described page 21 with the following results :—

TABLE VI.

Class.	Material per cent.	Assay of Originals.		Assay of Residues.		Hours' Contact with KCy.	Remarks.
		dwt.	grs.	dwt.	grs.		
Coarse, . . .	15	11	3	4	13	72	Ton—2000 lbs.
Medium, . . .	40	4	3	1	3	50	
Fine, . . .	34.8	9	1	0	18	50	
Slime, . . .	10.2	7	10	1	0	16	Percentage of slime found by difference.

In the first three cases the ore was treated by percolation, and the time given includes draining. The slime was treated by agitation, and includes time of settling and decanting.



Here we have 15 per cent. or 300 lbs. of coarse particles to re-crush to medium size for 3 dwts. 10 grs. per ton; and to the fine size, for 3 dwts. 19 grs. per ton; while if all was turned into slime, we should get only 3 dwts. 13 grs. per ton, and at an additional cost.

It was next decided to continuously re-crush the 15 per cent. of coarse material and return it to the classifiers to be sized with the original ore. This left only three sizes to be treated by cyanide, when the results obtained were as follows:—

TABLE VII.

Class.	Material per cent.	Assay of Originals.		Assay of Residues.		Hours' Contact with KCy.	Remarks.
		dwts.	grs.	dwts.	grs.		
Coarse, . .	15	10	19	...	...	...	Percentage assumed.
Medium, . .	38·8	4	12	1	4	50	
Fine, . . .	36	9	7	0	19	50	
Slime, . . .	11·4	7	9	0	22	16	

Thus, in this case it may be seen that it was a decided advantage to size and re-crush the coarse particles and distribute amongst the three other sizes.

It is noteworthy that the coarse particles from the small classifier are not usually so liable to crush to slime as the original ore. The increase in the quantity of slime was only 1·2 per cent., so that the increase in cost of treatment from this source is immaterial. The chief additional cost is in the re-crushing and re-handling, but this cost per ton, with a well arranged plant, should not exceed the cost of crushing the original ore per ton.

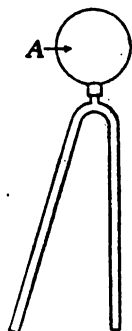


FIG. 10.

**Agitation Test.**—Agitation is chiefly employed for the purpose of treating slimes; and as the cost of treatment by this method is greater than by percolation, our aim in crushing is to produce as small a quantity of non-leachable material as possible. There are, however, cases where the whole of certain rich refractory ores have been treated more advantageously by agitation than by percolation.

A suitable plant for laboratory tests is described and illustrated on page 16 and in fig. 6. A usual method of procedure is as follows:—

The ore having been neutralised, as described on page 31, the cyanide is added gradually until the strength remains nearly constant (say 0·007 to 0·01, and contains about two to four parts of solution to one of dry slime). In treating

slimes in the laboratory, agitation of two to four hours is generally sufficient, and samples are taken at intervals of say 2, 2½, 3, and 3½ hours; these are washed with a superabundance of air-free water, filtered, dried, and ore assayed. The samples should be taken from near the bottom of the tank without stopping agitation. This is conveniently done by means of a syphon shown in fig. 10. One leg of the syphon is inserted into the mass of ore and the rubber ball *A* is squeezed. The end of the other leg is covered with the finger and kept air-tight, when on loosening *A* the pulp rises, and on removing the finger a steady flow escapes. The first portion taken should be returned to the tank.

Coarse or medium ore is treated in the same way as slime, except that it may require stronger cyanide solution.

Table VIII. gives the results of treatment of a sample of mispickel slimy sand that could not be treated by percolation, and presents a convenient method of tabulating the figures.

Weight of ore taken, 1000 lbs. dry.

„	water 400 lbs.	= 800 lbs. per ton ore.
„	lime 3·6 lbs.	= 7·2 lbs. „ „
„	caustic soda 0·1 lb.	= 0·2 lb. „ „
„	potassium cyanide 0·2 lb.	= 0·4 lb. „ „

TABLE VIII.

Assay of original ore, 6 dwts. gold, 3·3 dwts. silver.

Hours' agitating.	Per cent. KCy.	Assay of Residues in dwts.		Gold extracted per cent.	Remarks.
		Gold.	Silver.		
0·25	0·04	4·25	2·9	29	KCy decomposed 0·04 lb. Added 0·04 lb. KCy.
0·5	0·05	3·41	2·32	43	
0·75	0·05	2·9	2·24	51·7	
1	0·05	2·32	2·25	58	} Per cent. KCy. indefinite. Added 0·04 lb. KCy.
1·5	0·05	1·9	2·2	68·3	
2	0·05	1·62	2·2	73	
3	0·05	1·5	2·21	75	
4	0·05	1·52	2·2	75·3	
5	0·055	1·2	1·95	80	
6	0·055	1·2	1·95	80	

The cyanide used for 3 hours' treatment = 0·24 lb. = 0·48 lb. per ton ore.

„ „ 5 „ „ = 0·28 lb. = 0·56 lb. „ „

From the above experiment it is evident that a six hours' treatment is uncalled for, and it seems improbable that longer than two or three hours' treatment would be desirable. The difference in the extraction between three and five hours is only 0·3 dwt., or about 1s. 3d. (30 cts.) per ton. Then, again, there is only 0·12 dwt. or 6d. (12 cts.) extracted between two hours' and three hours' treatment. It must also be taken into account that time has to be

allowed for separation of the solution from the solid particles by filtration, or settling and decanting, and while this operation is proceeding the metal is still being acted on by free cyanide.

Since the addition of 0.04 lb. of KCy at the end of the fourth hour caused a higher extraction, it might be beneficial to add it earlier, say at the end of 1.5 hours, and then finish the treatment at the end of two hours. The time of treatment must depend very much on the cost of power and a number of other factors, which vary with local conditions. Experiments of this class have to be very varied and exhaustive to get a maximum extraction at a minimum cost.

It is noteworthy that the proportion of solution to ore was 2 to 5, but if a larger amount of solution had been added, so as to make the pulp thinner, the time necessary to get an equally good extraction would be less. This is a matter that will be dealt with under dissolution of gold and silver in slimes.

## CHAPTER V.

### CRUSHING TO CYANIDE

THE object aimed at is to break the ore into particles of such size as will leave the gold and silver capable of coming into contact with the cyanide solution. This end is attained by fine pulverising, and much may be learnt by examining the grains under a low power microscope to determine the minimum degree of fineness necessary; a point which, however, must ultimately be decided from empirical results by testing with cyanide solution. The more finely the ore is crushed the more perfectly does the gold and silver dissolve, but if crushed too fine an excess of slime or dust forms, which interferes with the separation of the dissolved gold and silver and causes an increase in the consumption of cyanide. If crushed too coarse, a large percentage of the gold and silver remains locked up and is not dissolved, but that portion which does dissolve may be separated, more or less readily. Under no circumstances can we crush an ore to one uniform size, but our aim should be to crush so as to get a maximum number of those particles that give the highest extraction at the lowest cost. This can be found only by experiments.

The size to which an ore has been crushed for the best results naturally varies with the character of the ore, and with the minerals which are intimately associated with the precious metals. Thus at the Mercur mine, Utah, where the ore is of a porous nature, particles of half inch or larger are treated for the best results, while in Western Australia the sulpho-tellurides are crushed to a fine powder in order that the gold may be dissolved out by the cyanide. Sometimes oxidised ore mined near the surface disintegrates or cleaves readily along lines containing the precious metals, in which case only very coarse crushing is necessary to give the most economical results. It is, however, desirable in most cases to crush so that the whole of the ore will pass at least a 30-mesh screen, and produce as small a quantity as possible of slime or dust.

Then, again, in many cases it is advantageous to crush wet, while in other cases dry crushing is found more beneficial. Local conditions and the nature of the ore have chiefly to do with the choice of methods, and it is only by systematic experiments on a commercial scale that the most economical means and the best degree of fineness can be determined.

**Dry and Wet Crushing.**—The advantages and disadvantages of dry and wet crushing have been discussed at considerable length by many able authorities, since the inception of the cyanide process. Perhaps too much has been said by the advocates of each system to disparage the other.

The chief points for comparison are—

1. First cost of plants per ton capacity.
2. Total cost of crushing, including interest on cost of plant and redemption.
3. Percentages of slime or dust produced.
4. Costs of sizing.
5. Losses through dusting or sliming.
6. Percolation and leaching of products.
7. Consumption of cyanide.
8. Facilities offered for controlling operations.

1. The cost of rolls or ball mills per ton capacity is probably lower at every gold-mining centre than stamp mills and wet crushing. But when stamp mills crushing wet and dry are compared, the former are undoubtedly the cheaper. Of course, where small and uncertain bodies of ore are to be dealt with, the factor 'capacity' is not necessarily taken into account. The first cost becomes then the item of chief importance, and ball mills with some simple form of dryer are generally found preferable.

2. There is a consensus of opinion amongst mining men, experienced in both methods of crushing, that the total cost per ton crushed is less with rolls or ball mills where the ore can be cheaply dried, but where the ore is wet and fuel is expensive the item for drying the ore may be so great as to make dry crushing prohibitive.

3. In crushing dry, the percentage of dust incapable of being leached in tanks in the ordinary way may exceed the percentage of slime produced in crushing the same ore wet, but the reverse is often the case. We have known an ore crushed dry in a ball mill, and when sized in four spitzlotten the unleachable slime passing away equalled 12 per cent. of the original ore, whereas when the same ore was crushed wet in a stamp mill the unleachable slime was about 9 per cent. In both cases the screens were the same size, viz., 30-mesh, but with the dry mill, when 20-mesh screens were used and the coarse particles re-crushed through a 30-mesh, the amount of slime diminished and the output increased. It seems probable the same thing would have happened with a stamp battery crushing wet.

In another case, where an ore was crushed wet through a 30-mesh screen in a stamp mill with a discharge three inches high, the slime produced was about 11 per cent., whereas the same ore crushed by the same stamp mill with a discharge twelve inches high produced a product which contained 38 per cent. slime. The results obtained from a two months' trial in each case are tabulated in Table IX.

From this it will be seen that in the attempt to get a higher extraction by use of a chock block in amalgamation the percentage of slime was largely increased, and also enriched with gold, showing, in fact, for an increase of 22 dwts. of gold per 100 tons crushed, the increase of gold brought into the slime was 182.2 dwts. Here we are likely to find a net loss, apart from the increased cost of crushing, as the profit on slimes is usually less than on sands. There was, however, in the first case about  $2\frac{1}{4}$  tons of coarse material to 1 ton in the

TABLE IX.

Discharge above Dies.	Tons crushed per Stamp per Day.	Slime percent. of Ore.	Gold amal- gamated per Ton.	Assay of Slime.	Gold amalgamated per 100 Tons.	Gold in Slimes per 100 Tons Ore.
			dwts.	dwts.	dwts.	dwts.
3"	5.6	11	5.7	5.2	570	57.2
12"	3	38	5.92	6.3	592	239.4

second case taken from the first spitzlutte. This required re-crushing to get the best results by cyanide.

4. Particulars of comparison between the cost of sizing dry and wet are wanting, but it is improbable that in well arranged plants, in either case, the cost would be a serious item.

5. Even in the best designed plants, when handling the ore, drying, crushing, roasting, or sizing, either with the dry or wet methods of crushing, unavoidable loss always occurs. Loss by sliming in the wet process is in most cases greater than the dusting in the dry, but the assay value of the dry dust nearly always exceeds that of the wet slime. This is a matter of importance which has to be gone into in drawing the comparison, as the actual loss in precious metal varies very widely with the plant adopted, and to some extent with the ore. The loss of gold and silver from this cause is generally approximately proportional to the quantity of pyrites or other metallic mineral matter in the ore, and not to the assay value of the ore. It is well known that in dry crushing, the finer the dust that escapes the higher is the assay value, while with wet crushing this is seldom the case.

6. When an ore is crushed wet and sized, clean leachable products may be obtained, which have been washed in the process free from soluble 'acid.' The consumption of cyanide is thus reduced to a minimum, and the solution flows easily through the whole mass, wetting all the particles evenly, but the moisture in the ore has the effect of decreasing the strength of the solution. When the ore is dry crushed the solution is not weakened in the same way, but soluble acid is retained, which may appreciably increase the consumption of cyanide. If crushed dry and treated unsized, a difficulty is often experienced in wetting the particles evenly. This may be reduced to a considerable extent by first sizing.

Many ores crushed dry in a moist atmosphere have to be treated immediately, as ores are generally hygroscopic, and readily oxidise the sulphides, etc., forming basic salts that react with cyanide. This action may, however, be retarded by spraying the ore, as crushed, with lime water, dilute alkali, or cyanide solution until it contains about 4 per cent. moisture.

7. Some ores, when crushed dry, consume such an excessive quantity of

cyanide that washing before treatment has to be resorted to. When this is the case, it is preferable to size the ore wet. By crushing wet, the washing is performed in the one operation. On the other hand, there are ores that when crushed wet or dry cause no appreciable difference in the consumption of cyanide. With wet crushing, the cyanide solutions often increase beyond the capacity of the sumps, and a portion has to be run away, which means a loss in cyanide, and possibly of gold also. In cyaniding dry crushed ore this does not occur.

8. One of the great advantages of dry over wet crushing is the facility it offers for controlling the weight of material treated, and the sampling. The weight is usually determined from bulk; and when an ore is dry, the weight by bulk remains practically constant for the same sized particles, but when wet, the weight by bulk may vary considerably, even as much as 40 per cent. Thus a certain ore that occupied only 19·6 c. ft. to the ton when dry, took, when moistened with 4·8 per cent. water, 27 c. ft. to make a ton after allowing for moisture. In another case, an ore that required 20·4 c. ft. to the ton when dry, required with 7 per cent. moisture 24·2 c. ft. When, however, an ore is crushed wet, sized, and charged direct into the tank, the weight by bulk remains practically constant for each size. In sampling wet a great deal of trouble is experienced, but in the case of dry ore the sampling may be conducted with great ease and accuracy.

We cannot have one ideal system of crushing, because we have so many different conditions to meet. What would be an ideal system in one locality may be quite unsuitable, even on similar ore, in another locality; and it too often happens that a system which was highly suitable in one district in the case of one class of ore, is blindly followed in another without regard to a change of conditions. Much money might often be saved and disappointment averted by the erection of several small plants of the most likely systems, to give the best products for cyaniding before adopting any particular system of crushing on a large scale.

Perhaps the ideal system of crushing to meet any given set of conditions will be found in a combination of both wet and dry methods, and in a greater combination of different appliances than are usually employed. The two processes of crushing to amalgamate and crushing to cyanide are so antagonistic to one another, that one method must, to some extent, give way to the other. That is to say, either crushing to amalgamate or crushing to cyanide must be selected as the chief aim of the mill man. If his aim is to get the gold chiefly by amalgamation, then fine crushing is generally essential for the best results. This means a small proportion of leachable products for cyaniding and an excessive quantity of slime. If, on the other hand, his chief aim is to get the most suitable product for cyaniding, then amalgamation must be either abandoned, or a smaller quantity of the gold extracted by this method must be accepted, and the cyanide relied upon to extract the balance. In the latter case, it is generally advantageous to crush the ore in several operations. In fact, it may be taken as a rule that the more times an ore passes into and

out of a crusher, being sized after each operation, the more even in size is the resulting product as a whole, and the better is it for percolation.

The number of combinations of different crushers, each having its use, are very great, but the following system is given as a suitable method of procedure when amalgamation is employed, and crushing to cyanide is the chief object. Let the ore, as it leaves the mine, be dumped into a crusher of the Edison rolls type, or of the type of the Gates crusher, and then pass on to sets of corrugated rolls, until it is reduced in size to pass a screen of  $\frac{3}{8}$  to  $\frac{1}{2}$  inch. In this condition the ore is weighed and sampled. These two operations might well be conducted with automatic weighing and recording apparatus and a sampling machine, and all extraction results, whether from amalgamation or cyanide processes, should be based on these weighings and the assays of the samples. The moisture determinations would of course be got from the samples taken for assay, and deducted from the gross weighings. In this way the management could have a check on operations which would be more satisfactory to all concerned than the loose methods usually carried on. Any irregular variations between theoretical and actual extractions could then be investigated, by which the cause might be found.

The ore, after having been sampled and weighed, is passed on to a fine crusher, preferably a stamp mill of a heavy type, giving a short, quick drop, and having a discharge as low as practicable. The mesh of the screen would naturally vary with the ore, and is best determined by experiment, but it is improbable that it would ever be advisable to use a screen of narrower mesh than 0.03 inch (24-mesh), to get a minimum of slime and a maximum number of ore particles of the size that gives the best extractions by percolation. The crushed ore, as it leaves the mortar box, is passed over wide amalgamated plates, with fall adjusted to suit the ore, and the flow of water to suit the fall, and the pulp then flows on to a set of sizers. It is generally found that the ore from the first sizer, and possibly from the second, is too coarse to give a high extraction, and will pay to re-crush in some suitable auxiliary crusher, such as a Huntington mill, or a ball mill for wet grinding. The re-crushed ore should be returned to the amalgamated plates and again to the sizers, the operation being continuous. By employing such a system and making amalgamation of only secondary importance, the volume of water used in the mortar box may be reduced, and so regulated that the crushing effect of the stamps and the number of particles forced through the screen by each splash is brought up to a maximum. This is of rather thicker consistency than can be streamed over the plates, and the extra supply of water for this purpose is supplied outside the mortar box. In this way the production of slime may be reduced to a minimum, as the readily slime-forming portion of the ore is removed from the mortar before it can be further reduced in size, and the hard, coarse particles, by being re-crushed separately, do not produce as much slime as the original ore (see page 36).

When an ore is to be crushed dry, it is advisable to first reduce it in size to  $\frac{3}{8}$  or  $\frac{1}{2}$  inch cube, sample and weigh as above described, and then dry in



some suitable dryer. The drier the ore is, the smaller is the production of slime in dry crushing and the better is the quality of the product for sizing and leaching by percolation. Care, however, should be taken not to dry the ore at too high a temperature, as some of the mineral constituents are apt to become oxidised and form basic salts, which have an injurious effect on the cyanide solution. As in wet crushing, so in dry crushing it is advisable to fine-crush the ore in more than one operation. A coarse screen may be employed in the first operation, and the particles which are too coarse to get a good extraction are then separated and re-crushed.

Rolls have been employed for dry crushing ore to cyanide, and in certain cases a fairly good product has been obtained. Stamps are sometimes used, but from the very nature of the operation and the difficulty of removing the crushed product as quickly as it is formed from the mortar box, dry stamping of ores to cyanide is not a desirable way of crushing. Ball mills of the Krupp-Grusonwerk type are generally preferable to either of the above methods. They should be of a fairly large size, and for the first crushing have coarse screens, 12- to 20-mesh being suitable in most cases. The product from this is sized, and the particles which are too coarse to cyanide are re-crushed in a separate mill with a finer screen. In this way several sizes may be obtained without forming an undue quantity of slime, as is often the case when the ore is crushed in one operation through a fine mesh screen.

It is sometimes desirable to crush an ore to as fine a slime as practicable. Thus, in the case of West Australian telluride ores when treated raw, special mills are employed for this purpose. The most suitable appliances have been found to be grit mills, similar to those used in Portland cement works. For ore particles that are to be ground wet, a mill of a modified form has been devised of the Krupp-Grusonwerk type, which is largely used in Western Australia. Both dry and wet mills do the work very thoroughly, and form for the most part a very fine powder or pulp, but at a cost which is prohibitive, except in the case of rich ores and concentrates. They require 12 h.p. per ton output per hour to produce a pulp of sufficient fineness for slimes treatment, and are made in five sizes, to treat from two to seven tons per hour.

## CHAPTER VI.

### WEIGHING AND MEASURING.

**Weighing the Ore.**—In the cyanide process it is highly desirable to know the weight of the ore being treated, in order to be able to control operations and compare results between the actual and theoretical extractions from time to time. The most reliable way of getting the correct weight of an ore is to charge trucks and weigh on platform scales just before dumping into the tanks, taking at the same time from each load a sample for moisture determination. The total moisture is finally deducted from the net weight scaled. Such a method is not always convenient, and at best is expensive and requires much labour. An approximation is arrived at by taking the net weight of truck loads at intervals of, say, every hundredth, and assuming this to be an average. In some cases the tank capacity by weight is determined, and it is assumed that each full tank contains the same weight. But it has been shown that in the case of ores crushed wet and charged moist, the weight by bulk varies very widely. In the case of dry crushed ores, or when crushed wet, sized and charged direct, the tank capacity as a whole remains fairly constant.

To determine the weight of a tank charge by measurement, fill a metal or wooden box with the crushed ore having exactly one cubic foot contents, dry at a moderate temperature, and weigh. Make twenty or thirty similar determinations at different times, and take the average as the weight of a cubic foot, from which the tank capacity can be determined. To determine the weight of an ore charged direct, place in the tank at different parts and at different heights similar boxes of one foot cube, but made of perforated sheet iron. Dig out after the tank has thoroughly drained, sweep off the sides, level, dry, and weigh contents. Repeat at intervals or when any change occurs in the ore, and take the average weight of twenty trials as a close approximation from which the contents of the tank may be determined.

When the cyanide works is run in conjunction with the crushing mill, as it should be in all cases, the extraction results should be based not only on the tonnage determined by measurement from the tanks, but on the actual weight of ore crushed. There is bound to be some unavoidable loss through sliming or dusting and transferring, but it is the duty both of the mill men and cyanide men to keep this loss at a minimum. One should work in harmony with the other, and it should be to their common interest to report at intervals how small their loss is from this source.

**Continuous Weighing.**—Several attempts have been made to devise an automatic machine for the continuous weighing of the ore as it is delivered for

treatment. Thus a platform scale has been devised which automatically weighs and records the net weight of truck loads while in motion with rope haulage, but many practical difficulties were met with which interfered with the accuracy of the results.

Recently Messrs Denison & Son, of Leeds, have introduced a novel machine known as the "Blake-Denison Continuous Weigher." With this appliance the ore is automatically weighed, and the number of tons recorded on an indicator while it is being carried on a belt conveyor. The principle on which this is constructed may be understood from the following.\*

Let us take the case of an ordinary belt conveyor supported by equidistant rollers. With any three of these rollers the middle one supports half the total load between the two on either side of it. Thus in fig. 11, if  $A$ ,  $B$ , and  $C$  are rollers supporting the belt with its load, the middle roller  $C$  supports half the load between  $A$ ,  $B$ . If, now, we take mid points  $w_1$  and  $w_2$  and suspend roller  $C$  in the usual way of ordinary multiple lever platform weighing machines, the weight between these two points (which is half that on  $A$ ,  $B$ ) is carried by  $C$  and may be recorded. It is then only necessary to arrange the

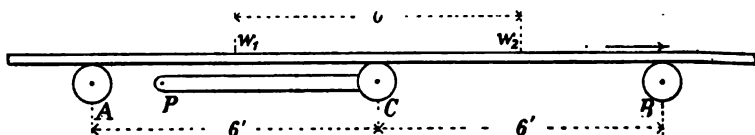


FIG. 11.—Blake-Denison Continuous Weigher.

mechanism so that it will record weighings of every length  $w_1$   $w_2$  (6 ft. in this case) of the belt while running to get a continuous, or rather a rapid, intermittent weighing machine.

It is obvious that the mechanism must be arranged to suit the speed of the belt, for if in the above case it travels at a speed of 3 ft. a second, it will require separate weighings recorded every 2 seconds, while if it is running at the rate of 4 ft. a second, it would require weighings recorded every  $1\frac{1}{2}$  seconds. The Blake-Denison weigher records these weighings on a counter which indicates in figures the number of net tons and decimals of a ton.

There is, however, a limit to the speed at which these weighings can be made with accuracy, and this is stated by the makers to be 350 ft. per minute. It is also said to weigh with equal accuracy when the belt is made to elevate the load, and this was tested when the angle was 1 in 10, with the result that it worked perfectly.

Several ingenious details are introduced into this machine; and as the mechanism is somewhat delicate, it is desirable to keep it housed in a dust-proof case. A glass window may be inserted in the case to allow the indicator to be read without opening.

The belt conveyor offers excellent facilities for taking samples automatically for assay and moisture determination as the ore is delivered. If the moisture

\* *Colliery Guardian*, Jan. 17, 1902.

is fairly constant, the weigher can be adjusted to record the net weight of dry ore.

For convenience in calculating, the short ton of 2000 lbs. is mostly employed at gold and silver mining centres, but at some places the long ton of 2240 lbs. is still taken. Some confusion exists in the use of the two different units, and it is to be hoped that the time is not far distant when one unit will be universally employed.

**Weighing the Solution.**—It is usual to refer to the quantity of solution applied, by the ton weight. A cubic foot of working cyanide solution usually weighs about 64 lbs., and from this the contents of a tank may be readily determined. The area of the solution tank in square feet multiplied by 0.32 gives the short tons per foot deep, and by 0.0266 the short tons per inch deep, or multiplied by 0.0285 gives the long tons per foot deep, and by 0.00238 the long tons per inch deep. The weight of solution run on the tank is conveniently determined by employing a measuring staff marked off in tons, or in any suitable number of tons according to the size of the tank. The staff is first dropped to the bottom and the tons contents noted, then the requisite number of tons are allowed to run into the leaching tank. The time of the day and the quantity should be immediately recorded in a book kept for the purpose. Such a method is sufficiently accurate for all practical purposes, but it has the objection, that, since cyaniding is carried on night and day, the management cannot have a check on the operations and see that they are carried out as recorded. Irregularity in this matter is sometimes accountable for the erratic results so often obtained.

It would be more satisfactory to use a simple recording meter that would automatically record the number of tons used, and the time of the day when a solution is applied. Some makers are at present experimenting with meters of this class.

## CHAPTER VII.

### PERCOLATION AND LEACHING.

THE process of causing a liquid to pass through a porous medium, such as sand, etc., is termed *percolation*, and when the liquid, as it percolates, dissolves some of the porous medium, it is said to *leach*.

In the cyanide process ores are treated almost entirely by percolation and leaching in tanks, and it is the exception to have to resort to other means, unless the ore is in the form of slime. The advantages of the percolation method of leaching in tanks are, that enormous quantities may be dealt with in one operation, and by the simplest possible plant, also, the method is easy to control in carrying out. Like all simple processes there are, however, many practical points to be observed that require some consideration, in order that the results may not vary widely.

**Rate of Percolation.**—This varies with the volume and uniformity of the interstices between the ore particles, and to some extent with the depth of the column, the pressure, and the temperature. The constituents of the ore also have some influence on the rate of percolation, for minerals such as galena, pyrites, and blende have a very marked effect in diminishing the rate of percolation, and this varies with the proportion of these minerals to the siliceous particles present. It naturally follows that the coarser the ore particles are the larger will be the pores or interstices and the more rapid the rate of percolation. Then, again, uniformity in the size of the ore particles has something to do with the rate of percolation and the leaching effect, for if an ore be crushed to pass through an ordinary assayers' sieve of 90-mesh, and washed free of slime, a good leachable product may be obtained, but if this fine ore be mixed with coarser particles, percolation becomes slower than the average of the two sizes treated separately, and leaching is retarded.

Percolation is carried on in either a downward or upward direction, and we have made some investigations on the rate in both directions. When downward, the rate does not vary very much with different depths of ore, but there is always a diminishing decrease with increase of depth. Thus an ore through which a solution percolated the first foot at an average rate of 1.6 inches per minute, percolated two feet at 1.3 inches per minute, three feet at 1.22, and four feet at 1.2. With upward percolation the average velocity for the first foot from the filter cloth was 8 inches per minute, for two feet was 4 inches, for three feet was 2.75 inches, and for four feet was 1.4 inches per minute, showing a rapidly diminishing velocity as the solution percolates towards the surface, caused chiefly by the difference of hydrostatic pressure. The level of

the solution was in both cases kept constant at three inches above the ore surface, for the purpose of comparison.

The loss in velocity, as the solution descends into the ore, seems to be due in a large measure to the pushing forward of the air contained between the particles, and also to the skin friction between the solution and the ore. In its attempt to escape, some of the air moves in an upward direction and sometimes reaches the surface, but a very large number of globules are retained, as the pressure does not enable them to force their way upward and escape. These remain in the ore, balanced with the weight of solution above. Such a state of things has a very marked detrimental effect on the rate of leaching, and causes an excess of moisture to be left in the ore after draining. (With upward percolation these globules of air are not formed to the same extent.) Of course, the chief quantity of the air below the ore surface is pushed forward and escapes through the outlet pipe below the filter, and if the outlet pipe is not open a large volume of air bubbles upwards through the sand, producing channels and cavities, and affecting the texture of the ore.

**Texture of the Ore.**—Uniformity in texture of the ore is one of the chief secrets of good leaching, and to get uniformity in texture we should have uniformity in the size of the ore particles. Thus a carefully sized ore will give not only more uniform results but better extractions, and will retain less moisture than the same ore unsized. The texture of the ore is largely influenced by the method of charging the leaching tank. We have made some determinations as to the volume occupied by the same weight of ore of different sizes when charged direct with water, moist, and dry, and have tabulated the results as follows:—

TABLE X.

Passed Mesh.	Retained on Mesh.	Relative volume when		
		Direct.	Moist.	Dry.
30	60	100	140	96
60	90	100	144	91
90	...	100	152	96

Thus it will be seen that the ore charged moist is the most porous, that charged direct is next, and that charged dry is least. In practice it is usually found that the dry charge percolates better than when the ore is charged direct with water. This is owing to stratification of slimy matter in the latter case, which is difficult to avoid.

**Retention of Moisture.**—The amount of moisture retained by a leached ore depends on the size of the particles, the uniformity of texture, air pressure and temperature, and to some extent on the depth of the column.

We have made some determinations as to the quantity of moisture retained after leaching and draining in the ordinary way. The ore had been washed quite free from slime and was then sized with assayers' sieves. Table XI. gives the results.

TABLE XI.

Passed Mesh.	Retained on Mesh.	Moisture per cent. Ore retained.	Remarks.
30	60	4.5	The temperature was between 61° and 65° F. and the pressure below the filter was 1 lb. under the atmosphere.
60	90	7.4	
90	...	9.2	

These experiments were extended by adding various percentages of slime, carefully mixed in a dry state, to an ore which had passed a 30-mesh and had been retained by a 60-mesh. Table XII. gives the results.

TABLE XII.

Slime added per cent.	Moisture per cent. Ore retained.	Remarks.
1	3.7	The temperature varied between 58° and 64° F. while the experiments were being carried out. The pressure below the filter cloth was not reduced in any case.
2	8.8	
3	10.3	
4	12.5	
5	14.4	
7.5	17.6	

When the pressure below the filter was reduced by 10 lbs. per square inch, the results showed that the moisture retained was irregular, being sometimes greater and sometimes less than the above, but when reduced only 1 or 2 lbs. the percolation was very uniform, and the moisture retained was always less than the above. After completing the leaching by gravity alone, the percentage of moisture was always decreased still further, by diminishing the pressure below the filter.

Temperature has some effect, which in many cases is marked. The higher the temperature, the smaller is the quantity of moisture retained.

A deep tank has a slight advantage over a shallow one in the quantity of moisture retained after leaching. There are always a few inches at the bottom of the tank that retain an excessive amount of moisture, and the depth of this, which is usually 2 to 6 inches, varies with the nature of the sand, but is independent of the depth of the column. When globules of air or hollows are

*in situ* in the tank of ore, the portion immediately above the hollow always retains an excess of moisture, just like that at the filter cloth. This is hardly affected by a reduction of pressure under the filter, and accounts for an undue amount of moisture left in a tank of uniform texture.

**Influence of Temperature on Percolation.**—A high temperature has a very marked effect in accelerating percolation, because the chief resistance offered to a solution in passing through the ore is internal friction, and this is reduced by increase of temperature. If the internal friction at freezing point is 100, at boiling point it would be only about 10 to 15.

**Influence of Pressure on Percolation.**—Gravity usually supplies the working pressure, assisted by a slight suction from the dip of the outlet pipe. Sometimes, however, a vacuum pump is employed, whereby the pressure below the filter is greatly reduced, but this method has drawbacks, and is not found desirable except in special cases. When the solution is forced through a tank of ore at a pressure near to that of the atmosphere, it is often found that the sand packs and the pores become somewhat choked, whereas if the applied pressure is only slight—about 1 to 3 lbs. per square inch—this does not occur appreciably. Moreover, in the final draining the higher pressure tends to leave a larger quantity of moisture behind. This seems to be due to the solution descending too rapidly to allow the globules held between some of the particles to overcome the internal friction, and the air then surrounds these globules and prevents them agglomerating. This is a case of capillarity, and is most apparent in ore loosely charged. If, however, the ore has been drained by gravity alone, the use of a vacuum pump or other suitable appliance at the end of the operation will, as already stated, usually draw off more moisture.

When a pressure higher than that of the atmosphere is required to make the solution percolate, it is usual to resort to the filter press. Highly satisfactory leaching may be obtained by this method, but the additional cost of plant and handling may make the process too costly to work, except in a few cases. In West Australia filter presses are largely used to leach and dry the slimes of rich telluride ores.

In America, strong covered cylindrical tanks have been employed on ores difficult to leach under atmospheric pressure. The solution was pumped in on the top of the ore and a high pressure of air—two or three atmospheres—was applied, which forced the solution uniformly down to the filter bottom. The high pressure causes the solution to become surcharged with oxygen, and thus aids dissolution of the gold. The process is too expensive to be applicable to any but exceptionally rich ores.

**Causes of Bad Leaching.**—Bad leaching is chiefly the result of uneven percolation, and uneven percolation follows from want of uniformity in the texture of the ore, or from want of uniformity in the pores of the filter cloth. It often happens that a tank is charged with an ore containing an excess of moisture, and a good deal of fine slimy matter gets to the filter cloth, some passing through, and some remaining and choking the pores. When this happens, the extractions



in that tank become poor, and the cause is often not observed until a considerable loss has occurred.

It therefore becomes incumbent on the cyanide operator to wash the filter cloths often, and take every precaution that each charge is of uniform texture, and that the ore particles are properly sized.

At some works it is usual to cover the filter cloth permanently with clean coarse sand a few inches deep. This helps very much to prevent choking and keeps the cloth clean. The sand can be removed and renewed at intervals of one month or so. An open loose grating may be placed 3 or 4 inches from the cloth as a gauge for the shovels in discharging the tank. The bottom 3 or 4 inches of sand is not removed with each discharge of the tank.

It often happens that bad leaching is due to the ore packing in places from vibration. When shafting, or a pump, or the staging used in charging is attached rigidly to the leaching tank or some place that allows it to vibrate, the effect is to cause the ore at that part to settle down, and percolation in the tank takes place unevenly.

**Methods of Applying the Solutions.**—The question may be asked—How can one best apply the solutions to the ore so as to obtain the greatest profit? Like many other questions it is impracticable to answer, because there are such a number of factors to be taken into account, many of which vary with different ores and in different localities. With a careful practical investigation by a competent man, he can, however, arrive approximately at a solution to the question for any particular locality, but different men may obtain the same results in different ways.

The solution is applied to the ore at the surface and allowed to percolate downwards by gravity, or it is applied to the ore from below the filter cloth and percolates upwards by hydrostatic pressure until it rises to the surface. The rest of the solution is then usually run on to the top of the ore and is drawn off by gravity.

The solution is applied continuously or intermittently. The former method consists in adding each solution soon after the previous one has sunk below the surface of the sand, without waiting for the ore to drain. Each succeeding solution is then said to displace the former, and percolation is kept up continuously. By the intermittent method each solution, after it has been allowed to stand for a time, is drawn off, and the ore is drained as well as time will allow, and then the next solution is applied. Some operators apply the solutions in three or four large charges, while others apply them in a good many small charges. Theoretically the latter method should give the best extractions, but in practice the former gives equally good results.

It is in the final treatment, when the water wash is applied, that a number of small charges are highly beneficial. Three or four small charges of 10 tons each will extract more of the salt adsorbed by the ore than one large charge of 30 or 40 tons. This is particularly noticeable in the first small charge of water wash, which after percolating will often be found, when drawn off, to be richer in alkali, cyanide, and gold than the previous weak solution. The cause

of this is not clear, but it would appear to be due to some action between the adsorbed salt and the air drawn into the ore mass as the solution drains off, whereby the adsorbed salt becomes more soluble.

The addition of a solution of a salt instead of a plain water or dilute cyanide as the final wash would in some cases be beneficial in preventing colloidal precipitates from being broken up and carried through the filter, which so often occurs. The salt causes the precipitated colloids to remain coagulated, and should of course be one that would not react with the cyanide.

The final wash to the ore is one that deserves more attention than is usually given to it, for after the gold has been dissolved a considerable proportion often remains behind in solution. This is held chiefly by adsorption and capillarity, and the amount removed by each successive wash is a rapidly diminishing quantity. This points to the difficulty of removing the metal even after dissolution, as the number of washes and the quantity of solution capable of being applied in practice is limited by time.

To determine the effect of water washes, we saturated a column of clean siliceous sand 100 lbs. in weight with 25 grs. of gold as  $\text{KAuCy}_2$  in a solution of weak cyanide. We next took a weight of water equal to the weight of sand and divided it into six equal parts. Each sixth was applied separately and allowed to drain completely, and the gold in the sump solutions was determined. Table XIII. gives the results.

TABLE XIII.

No. of Wash.	Gold in Sump Solutions, grs.
1	16.6
2	4.11
3	1.3
4	0.5
5	0.22
6	0.07

This shows how rapidly the quantity of gold salt washed out by each wash diminishes, and that after the third wash the amount separated becomes very small. It is problematical whether, with working conditions, it would pay to give the fourth or fifth charge, and the sixth would certainly not be profitable. The gold in solution, in this case, is about ten times as much as would be present in ordinary practice. This is simply to reduce the effect of experimental error.

An identical sample of sand was saturated with gold solution exactly as in the last case and was washed out with water, but in one charge, equal to the weight of the sand, when the total gold separated was 19.8 grs. In the first case the total gold separated was 22.7 grs., showing a better extraction by 2.9 grs. There was, however, in the first case 3.3 grs. and in the second case 5.2

grs. of gold left in the sand. There seems to be room for improvement here, and from experiments we have made it seems likely that if the final charge of wash waters contained a suitable acid or salt, the last trace of dissolved gold would be more completely removed.

**The Effect of Slime on Leaching by Percolation.**—We procured an ore in which the gold had been very uniformly distributed, so that when passed through a Clarkson Divider thirteen different samples did not vary by assay more than a few grains per ton, and on the average contained 16·6 dwts. gold and a little silver. The ore had, previous to assaying, been washed free from slime, so that when agitated with clean water and allowed a minute to settle only a slight turbidity could be seen. Twelve of the samples were mixed with different percentages of slime, varying from 1 to 12 per cent., and each sample was treated by the percolation method with strong solution 0·2 per cent. KCy, weak solution 0·05 per cent., and a water wash occupying 72 hours in each case. The slime added was plastic clay, free from gold and silver, containing 89 per cent. silica, the remainder being oxide of iron and alumina. The results are given in Table XIV.

TABLE XIV.

Assay Value of Originals 16·6 dwts.

Slime per cent.	Assay Value of Residues, dwts. Gold.	Extraction per cent.
0	2·6	84·3
1	2·77	83·3
2	2·93	82·3
3	3·4	79·5
4	3·6	78·3
5	4	75·9
6	4·48	73
7	4·94	70·2
8	5·5	66·9
9	5·6	66·3
10	6·39	62·6
11	9·3	31·9
12	...	...

With 12 per cent. slime the percolation became too imperfect to make the results of value.

A portion of the slime had been agitated with cyanide solution, and on testing the solution afterwards it was found to have lost only an inappreciable quantity of cyanide, so that the diminution in the extraction as the slime increased could not have been due alone to decomposition of the solution by the slime constituents.

It might be fairly safe to assume that the diminution in extraction was largely due to the ore particles becoming coated with slime, which hindered the solution from getting in contact with the gold and also to the difficulty of

washing out the dissolved gold, absorbed by the slime. It has already been shown how the presence of slime causes a retention of moisture.

**Quantity of Solution to cover the Ore.**—This evidently depends upon the amount of space between the grains, and will vary somewhat according to the nature of the material. The space between the grains varies with the size of the grains and with the amount of moisture present in the ore when the vat is charged, as already shown. The quantity of solution may easily be determined for any particular material by finding the specific gravity of the solid ore ( $S$ ) and that of the crushed materials \* ( $s$ ), then the proportion of spaces between the ore grains or voids =  $\frac{S-s}{S}$ .

To take an actual case of tailings from clean quartz so crushed that 1 per cent. was retained on 20-mesh sieve, and 5 per cent. on 40-mesh had a specific gravity of 1.61.† Assuming the sp. g. of solid quartz to be 2.65, then  $\frac{2.65 - 1.61}{2.65} = 0.392$  = proportion of voids. Therefore the proportion of voids to solids is as 0.392 is to 0.608, and the relative weights are 1.61 of sand to 0.392 water, or each ton of this sand will require 0.243 ton of water, or say  $\frac{1}{4}$  of a ton.

With carefully sized sand the specific gravity was found to be 1.28, the proportion of voids is therefore 0.52, and 1 ton of such sand requires 0.406 ton of water to cover it. It may be taken, then, that the quantity of water required in most cases is between  $\frac{1}{4}$  to  $\frac{1}{2}$  of the sand by volume, and between  $\frac{1}{4}$  to  $\frac{4}{10}$  of the sand by weight.

#### ADSORPTION PHENOMENA.

It is well known to analytical chemists that when substances are precipitated from solution, a portion of the precipitant is carried down by the precipitate. Thus alumina which is precipitated by alkali, carries down with it a large quantity of the alkali adhering to the alumina precipitate. The same property is possessed, to a greater or less degree, by the surfaces of particles of slimes, sands, and other solid substances in solutions of electrolytes. These phenomena are called *adsorption*.

Adsorption phenomena are developed to a considerable degree in the cyanide process, for we have to deal with an enormous area of contact-surfaces between the solids and the solution, and with media—silica, hydroxide of iron, etc., highly favourable for producing the effects.

Gore ‡ specially prepared pure silica, and in a number of cases he agitated

\* Specific gravity of crushed material =  $\frac{\text{weight of cubic foot}}{62.5}$ .

† Weight of a cubic foot of ordinary pyritic quartz sand, dry and free from slime, crushed wet through 900-mesh = 110 lbs. ∴ Sp. G. =  $\frac{110}{62.5} = 1.77$ . The same sand containing 5 per cent. moisture loosely charged = 82 lbs. ∴ Sp. G. =  $\frac{88}{62.5} = 1.33$ .

‡ *Proceed. Birm. Phil Soc.*, vol. ix. part i.

25 c.c. of different salt solutions of different strengths with 50 grains, in each case, of the silica. The supernatant liquid was then analysed and the results tabulated. Those in the case of KCy and NaCy are given below in Table XV., but we have calculated out the losses of cyanide in lbs. per ton of silica.

TABLE XV.

Original Solution KCy per cent.	Residuary Solution after contact with Silica 16-24 hours.		Original Solution NaCy per cent.	Residuary Solution after contact with Silica 16-24 hours.	
	Loss in lbs. per ton Silica.	KCy loss per cent.		Loss in lbs. per ton Silica.	NaCy loss per cent.
0·025	11·7	87·5	0·025	11·6	86·7
0·25	65·8	49·0	0·1	8·9	16·6
1·0	140	26	1·0	94·4	15·4

The result obtained with the 0·1 NaCy solution seems to be widely irregular. The loss of cyanide appears excessive when compared with the half to two pounds loss per ton in practical cyaniding, but Gore experimented with fine precipitated silica, which is a far more effective agent for showing the adsorption phenomenon than the sands usually treated by the cyanide process.

It would appear from the above results that the actual loss in cyanide is greater the stronger the solution, but the percentage loss is greater the weaker the solution, and this is what is found in practice. The influence of time of contact and temperature, between the limits ordinarily employed in the cyanide process, have little or no effect on the amount of salt adsorbed. A mixture of salts as usually found in working solutions produces a disturbing influence on the amount adsorbed, which becomes irregular. Other substances mixed with the silica, as alumina, oxide of iron, etc., may affect adsorption materially.

A freshly formed precipitate, as when an alkaline solution is added to an iron salt, so often occurring in the process, carries down with it a much larger percentage of the alkali than a similar precipitate previously prepared.

**Investigation of the nature of Adsorbed Substances.**—Linder and Picton and others have studied this subject, and have found that the solid particles actually decompose the electrolyte, carry down and adsorb the positive ion, while the negative ion is liberated and remains free in the solution. Whitney and Ober\* have extended these investigations and made accurate determinations on arsenious sulphide in colloidal state, using various salts as precipitants—barium, strontium, calcium, and potassium chlorides—and proved that the precipitate causes the salt to hydrolyse, carry down the

\* *Jour. Am. Chem. Soc.*, vol. xxiii. No. 11, Nov. 1901.

base, and leave the negative ion free in solution. Also, the base carried down and adsorbed was in proportion to the chemical equivalent weights of the barium, strontium, calcium, and potassium.

The same authorities give good reasons for asserting, what is doubtless true, that the fine solid particles cause hydrolysis of the salt, whereby the acid is left free in the solution, while the metal is carried down and adsorbed as a basic hydroxide.

In the case of working cyanide solutions the presence of impurities produces complications and causes very irregular results in the loss by adsorption. A slight excess of caustic alkali, as found useful in practice, has a marked effect in reducing this loss, while the presence of carbonates increases it.

With slimes and sands, such as one deals with in the cyanide process, the adsorption phenomena is not developed to such a high degree as in the case of the so-called colloids, but colloidal precipitates often occur in the treatment. Little or nothing, as yet, is known about the laws of adsorption, but it is generally believed that there is a proportionality between the surface-area of the solids and the amount adsorbed from the same solutions, and by solids of the same nature.

M. J. Thoulet \* made the following experiment to show that solid particles carry down air as well as salts in solution. A vertical tube is filled with boiled water, to which pyrogallie acid is added with a little gum arabic to retard the oxygenation. Fine grains of solid matter are then dropped into the tube, and as they reach the bottom each one is surrounded by a dark aureole, "showing well that it has fixed and carried down an envelope of air."

\* *Annales des Mines*, Experiments in Sedimentation, vol. xix. p. 1 *et seq.*

## CHAPTER VIII.

### PRINCIPLES INVOLVED IN THE DISSOLUTION AND PRECIPITATION OF METALS.

WE can do no more here than just touch on a few of the principles involved in these subjects, and for further pursuance the reader is referred to the numerous text-books\* which deal with the matter fully.

**Diffusion and Osmotic Pressure.**—If we take a vessel divided by a porous membrane into two compartments, both containing water at different levels, the two liquids will in time adjust themselves until they are both at the same level. If next we dissolve a substance in one compartment, such as cannot easily pass through the pores of the membrane, it will be found that part of the water in the other compartment forces its way into the solution of the dissolved substance, until both liquids remain at rest at different levels. This phenomenon is termed *osmosis*, and the pressure to which this change is due *osmotic pressure*. Osmotic pressure plays an important part in the cyanide process, both in the dissolving and precipitating of the gold and silver.

Prof. Pfeffer investigated this subject, and succeeded in preparing suitable membranes by which he made it possible to determine quantitatively the osmotic pressure of a number of chemical substances at different concentrations and temperatures. It has been proved in some cases that at the limiting surfaces between the water and the dissolved substance this pressure amounts to over a thousand atmospheres.

Van't Hoff, who had interested himself in Pfeffer's work, published a volume in 1887, showing that there existed a relation between the gas pressures of gases and the osmotic pressure of solutions. That is to say, that the well established gas laws of Boyle, Gay Lussac, Avogadro, and others also apply to the osmotic pressure of solutions. These laws are usually expressed by the equation

$$pv = RT \quad (1)$$

where  $p$  is the pressure and  $v$  the volume of the gas, while  $T$  denotes the absolute temperature and  $R$  is a constant for a perfect gas. There are, however, a number of exceptions in which the osmotic pressure does not obey the gas laws, and these are of most interest in the cyanide process. The exceptions include salts, acids, and bases, and in order to apply the above equation Van't Hoff introduced a coefficient  $i$  which modified the expression to

$$pv = iRT \quad (2)$$

If a salt such as potassium chloride be dissolved in alcohol, equation (1)

\* *Solutions*, by Ostwald, *Elements of Electro-Chemistry*, by Lüpke, and ditto by Le Blanc.

holds good, and the coefficient  $i$  does not come into account ; but if the salt is dissolved in water, the solution behaves as if a greater number of molecules were present than the concentration warrants. Thus, by halving the concentration, the osmotic pressure was always greater than half. It is concluded that this is due to a dissociation or breaking asunder of some of the molecules which occur in aqueous solutions, but not in alcoholic solutions.

**Electrolytic Dissociation.**—Clausius\* put forward a theory that when a salt such as NaCl was dissolved in water, some of the molecules broke down into ions ; that is to say, while some of the molecules remained NaCl, a certain portion, as it were, broke asunder and became Na and Cl ions, and these ions were free to move in all directions as if they were independent molecules. Now, if the osmotic pressure was greater than it should be in accordance with the gas laws, it would appear to point to a greater number of particles present than assumed, such as the theory of Clausius suggests.

The conception of the independence of certain parts of the dissolved molecules was up to this point only of a qualitative nature. In 1887 Arrhenius put forward methods in two distinct lines of research, by which he was able to measure what percentage of the molecules became dissociated, and offered an explanation which accounted for the exception in Van't Hoff's generalisation.

The one method was based on the electrolytic conductivity of the solution, and the other on the thermodynamic theory of osmotic pressure, both methods giving results that agreed within limits of experimental error. Also it was soon found that only those substances that show abnormally high osmotic pressures conduct a current. From this it may be inferred that only those solutions are conducting that are, at least in part, dissociated into ions.

Arrhenius called undissociated molecules inactive, and those which had broken down into ions active, and concludes that only active molecules or ions take part in the conductivity of the current ; and further, that when a solution was highly diluted, it became completely dissociated into active molecules or ions. These ions differ from atoms and molecules inasmuch as they carry enormous electric charges, the one, *e.g.* Na, being positive, and the other, Cl, being negative, whereas atoms and molecules are electrically neutral. NaCl dissolves in water as molecules ; but if this compound be decomposed in the presence of water into Na and Cl molecules or atoms, they instantly decompose the water and form new compounds. On the other hand, if NaCl is dissociated into ions, no such action takes place.

**Electrolytic Solution Pressure.**—Nernst conceived the idea that there was an analogy between the dissolving of a metal in a solution and the evaporation of a liquid into space. We are familiar with the fact that, at a given temperature, if a solution is evaporated it continues to pass into space until its vapour exerts a certain definite pressure called the vapour pressure, when equilibrium is established.

Now we may assume, according to Van't Hoff's theory, that the molecules

\* *Pogg. Ann.*, ci. p. 338, 1857.



of a dissolved substance exist under a definite pressure, which may be ascribed to the dissolving substance in contact with the solvent; further, that they have a power of expansion, being driven into space, where they exist under a certain pressure. Then, just as a liquid passes into space until the vapour in the space exerts a pressure equal to the vapour pressure, so a substance dissolves in a liquid until the osmotic partial pressures of the molecules equal the solution pressure of the substance.

Nernst put forward an explanation, based on his theory, to account for the action of primary cells, from which it may be concluded that the seat of electromotive force is chiefly at the surface of contact between the electrodes and the electrolytes. When a bar of zinc is immersed in an acid, some ions pass into solution, and in order to do so they must take positive charges with them. The metal then becomes negative, and the solution being positive from the ions it has received, we get formed at the plane of contact the so-called electrical double layer of Helmholtz, which resists any more ions coming into the solution. That is, the metal stops dissolving, although the amount dissolved is so small that it cannot be detected analytically. If, now, a negative metal such as lead be immersed in the same solution, it immediately becomes covered with the positive zinc ions, which charge it with positive electricity. This proceeds until a certain potential is attained, when equilibrium is established.

Let us next touch the zinc and lead momentarily. The positive electricity on the lead flows by the metallic conductor to the zinc, which had been negatively charged. The zinc being now positively charged, a fresh layer of zinc ions are again sent into the solution, which again leaves the zinc negative and charges the lead positively. If this operation is repeated a number of times it will be found that the zinc bar has lost weight, and that zinc has accumulated in the solution. In this way the metal dissolves, but if the two metals are kept in contact the process is continuous.

It is evident from Nernst's theory that the metal cannot send as many ions into a salt of zinc as into the pure acid, because the osmotic pressure of the zinc ions already in solution opposes any further ions being sent in, and therefore the solution pressure is lowered. Hence a metal becomes less soluble as metallic ions accumulate.

When the osmotic pressure of metallic ions in a solution is greater than the solution pressure of the metal, the metallic ions give up their positive charges and separate as atoms, *i.e.* precipitate. The solution now becomes negative, due to loss of some of its positive charges, and we again have the double layer at the surface of contact between the metal and the solution, but this time in the reverse order; that is, the solution is negative and the metal positive. There is therefore a similarity in the process of precipitation to that of dissolution.

**Anions and Cations.**—When a current is passed through a salt solution, that part where the current enters the solution is called the *anode* or *positive electrode*, and the part where it leaves the solution is called the *cathode* or

*negative electrode.* The ions of the salt are directed by the current and are caused to move, the negative ions or anions going to the anode, and the positive ions or cations to the cathode. This action of the current is used to determine what part of a salt is the anion and what part is the cation. For simple metallic salts the metals are the cations and the remainder the anions. In bases, the hydroxyl is the anion and the remainder the cation; and in acids, hydrogen is the cation and the remainder the anion. When, however, we have complex compounds, these simple rules do not apply. Thus, for instance,  $K_4FeCy_6$  dissociates into  $K_4$  as the cation and  $FeCy_6$  as the anion, and  $KAuCy_2$  into  $K$  and  $AuCy_2$ . Iron and gold in simple salts are always cations, whereas in complex cases they form part of the anions. These examples serve to show, what may be cations under one condition may be anions under another.

Ions (anions and cations) may carry one, two, three or more charges of electricity, and are usually, from this property, spoken of as univalent, divalent, trivalent, and so on. Ions of the same chemical composition do not necessarily carry the same number of charges, but may under one condition be univalent, and under another divalent, and so on. Thus in  $KMnO_4$  we have  $MnO_4$  as the anion, which is univalent, but in  $K_2MnO_4$  we have also  $MnO_4$  as the anion, which in this case is divalent. Also in  $K_4FeCy_6$  we have the anion  $FeCy_6$  carrying four charges, but in  $K_3FeCy_6$  the anion, although still  $FeCy_6$ , carries only three charges. Consequently the properties of the two compounds are different.

## CHAPTER IX.

### DISSOLUTION OF THE GOLD AND SILVER.

#### *Section I.*

It is well known that there are great differences in the solubility of different metals in the same solvent, and of the same metal in different solvents, but no satisfactory explanation has been put forward why these differences exist, and until this is done the process of dissolving metals must be regarded as imperfectly understood. Thus we know that platinum and iron are but slowly dissolved in potassium cyanide, whereas gold and copper are rapidly attacked. Also we know that gold dissolves in chlorine water, while it does not dissolve in nitric acid; and that silver dissolves in nitric acid, while it does not dissolve in chlorine water.

What really causes a solution to dissolve one metal in preference to another is difficult to explain, but to answer our purpose in discussing the solvent action of a cyanide solution on gold and silver, we assume the hypothesis to be correct that to dissolve a metal a difference of potential must exist between two parts in electrical contact with its surface and in contact with the solvent. It is well known that a pure metal is less soluble in a solution than similar metal of commercial quality, and we assume that if the metal is perfectly pure and has a surface of uniform density it would not dissolve in a solution of uniform concentration, because it would then have no difference of potential at any two parts. It would then be said that the solution pressure of the metal was the same at all parts of its surface; and in order that the metal may dissolve some alteration must take place, so that the potential at two parts of its surface may differ.

Two conditions are possible to produce a difference of potential between metals or metallic substances: (1) different substances, or the same substance of different density in electrical contact in the same solution; (2) the same substance of uniform density in different solutions, or the same solution in different states of concentration (strength).

The first case is the main source from which a difference of potential is derived in cyaniding the ore, by which gold is caused to dissolve and electric currents are produced.

Gold is never found in nature in a pure state, but is always alloyed or in intimate contact with other metals, and is generally in contact with some substance electro-negative in cyanide solutions to gold and silver, such as iron or arsenical pyrites. Thus if we take the case of a crystal of pyrites, *A*, fig. 12,

to which is attached a piece of gold *B* immersed in a solution of potassium cyanide, we find that the gold, being electro-positive to the pyrites, dissolves. The dissolving action produces electric currents, which are made possible only by having a negative electrode, *e.g.* the pyrites surface. The gold is therefore the anode of a simple cell, and the pyrites the cathode. The currents take an innumerable number of paths, as *a—b*, through the solid electrode, and thence through the solution, as shown by the arrows, the action being identical with that described under *Electrolytic Solution Pressure*, page 59.

In the second case we shall deal with a piece of gold not in contact with pyrites or other negative solid, but in contact with different solutions, or in solutions of different strengths.

If *A, B*, fig. 13, represent the piece of gold, of which the end *A* is in a strong solution of cyanide and *B* in a weak solution, we then have the part in the strong solution of a different potential from that in the weak solution. It

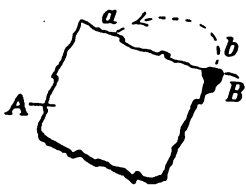


FIG. 12.—Auriferous Pyrites in a Cyanide Solution.

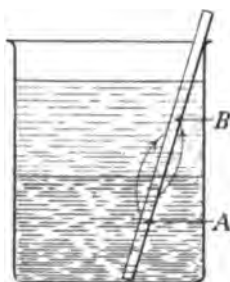


FIG. 13.—Gold in Cyanide Solutions of different Strengths.

will be found that the end *A* dissolves, and is therefore the anode, and the currents formed are led out of the solution at the end *B*, or cathode.

(The conditions in this case do not obtain, in treating ores by the cyanide process, to anything like the same extent as in the former case.)

(In both these cases we are dealing with short-circuited cells, identical with that formed in the process of dissolving gold and silver as contained in ores, but it would be impossible for us to make satisfactory investigations of the changes that occur while the two electrodes are in contact in this way. If, however, the anode and cathode are separated and short-circuited, by means of wires brought to the outside of the solution, with a suitable galvanometer intercalated, we have then a ready and accurate means at hand of noting electrical and chemical changes.

**Dissolving Investigations.**—In fig. 14 we have similar conditions to those in the first case cited above, except that the pyrites *A* and the gold *B* are separated and connected through a galvanometer *G* by means of wires conducted outside the cyanide solution. Given the same size electrodes, the chemical and electrical changes are obviously identical, except perhaps through an alteration in the resistance of the circuit, the rate of chemical action is

varied proportionately. When contact is made between the electrodes and the galvanometer, the deflection indicates that a current flows from *A* to *B* through the wires, and thence through the solution from *B* to *A*. The measurement of the current strength gives us data from which we can determine the quantity of metal dissolved, and compare results with any change in the conditions.

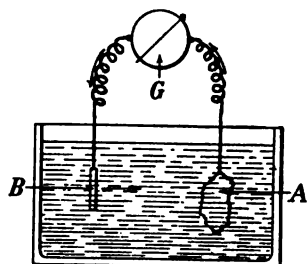


FIG. 14.—Gold in contact with Pyrites in Cyanide Solution.

**Electro-chemical Changes.**—It is assumed that the KCy in solution is ionised, either wholly or partly, into K cations and Cy anions, and that only ions take part in the dissolving action of the solution on the metal. When the current flows from the gold into the solution it takes positive charges, under the influence of the Cy anion that conducts the current into the solution, and negative electricity flows away through the connecting wires. At the same

moment K cations reach the pyrites, giving up their positive charges. The gold as attacked apparently forms AuCy, and this dissolved by the diffusion of an excess of KCy present to form KAuC<sub>2</sub>. The K atoms separated at the pyrites cannot remain in a free state in the solution, but instantly attack the water, liberating H and forming KOH. The hydrogen becomes occluded either wholly or partly by the pyrites electrode, and to some extent interferes with the further dissolution of the gold, as we shall see later.

**The Function of Oxidisers.**—The value of oxygen or oxidising agents has been much discussed, and although a few have disputed the necessity of oxygen for dissolving gold in cyanide solutions, the consensus of opinion is that some form of an oxidiser is essential for the successful carrying out of the cyanide process.

Elsner \* stated in 1846 that gold is only dissolved by a cyanide in presence of oxygen, and at a later date an equation was given in Watt's *Dictionary of Chemistry* to meet Elsner's views as follows:—



J. S. Maclaurin † has made some interesting investigations on the effect of oxygen in dissolving pure gold plates in a solution of potassium cyanide, and he has come to the conclusion that—(1) "Oxygen is necessary for the dissolution of gold in potassium cyanide, and no gold is dissolved in its absence"; (2) "The ratio of the gold dissolved to the oxygen required for its dissolution is 196 : 8, as demanded by the equation" cited above.

W. Bettel ‡ floated gold leaf on a cyanide solution and excluded oxygen by a continual passage of a stream of hydrogen, when no gold dissolved. He then tried the effect of the addition of different oxidising agents, as K<sub>2</sub>FeCy<sub>6</sub>,

\* *Erdm. Jour. Prak. Chem.*, vol. xxxvii. pp. 441–446, 1846.

† *Jour. Chem. Soc.*, vol. lxviii. pp. 199–212, 1895.

‡ *S. African Min. Jour.*, May 8, p. 666, 1897.

$\text{KMnO}_4$ ,  $\text{N}_2\text{O}_5$ ,  $\text{H}_2\text{O}_2$ ,  $\text{Cl}$ ,  $\text{Br}$ , etc., when the gold dissolved, more or less, rapidly.

J. S. Macarthur\* says, "I never could find that the presence of oxygen was necessary, either to dissolve gold by itself or from ores by cyanide." He says he has seen no evidence that oxygen is absorbed in the dissolving of the gold, and has no proof that hydrogen is evolved, but thinks the latter the more probable.

S. B. Christy† says that his investigations confirm the accuracy of the reaction attributed to Elsner, and asserts that "pure cyanide of potassium in pure water from which all other substances are excluded is entirely without action on metallic gold,"

We have made some investigations on this subject, dealing with working cyanide solutions from which oxygen has been extracted, and gold as it exists in ores in contact with pyrites. We find that—

1. The gold and silver are capable of dissolving at ordinary atmospheric temperatures and pressures to a limited extent, without the addition of oxygen or oxidising agents.

2. For the continued dissolution of gold and silver at ordinary atmospheric temperatures and pressures, oxygen, or an oxidising agent, is desirable if not absolutely essential.

3. At ordinary temperatures the atmospheric pressure has an indirect influence on the constancy of the dissolving action.

4. Temperature has a direct influence on the dissolution of the metal, independent of the oxygen absorbed by the solution.

5. The value of oxygen absorbed by the solution is dependent on atmospheric pressure, and to some extent on temperature.

The following experiments illustrate the action and confirm to some extent the office of oxygen. In fig. 15 we have a

similar apparatus to that already described. *A* is the pyrites and *B* the gold in a working cyanide solution, and these are connected by insulated wires to a galvanometer *G*. As has already been shown, a current is generated by this combination, flowing from the pyrites through the galvanometer to the gold, and thence through the solution to the pyrites again, and so on. Before making contact with *A* and *B* the galvanometer needle is brought to zero, and immediately after contact the deflection is noted. It will be observed that the needle returns again towards zero, at first rapidly and afterwards more slowly, until a point is reached when the needle remains nearly constant. Thus, in a case where the needle deflected to 20 divisions on the scale at the moment of contact, it returned in a few

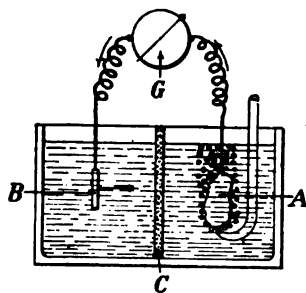


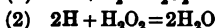
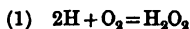
FIG. 15.—Gold and Pyrites in Cyanide Solution and the Effect of Oxygen.

\* *Jour. Soc. Chem. Ind.*, vol. ix. pp. 267–270, 1890.

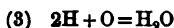
† *Trans. Am. Inst. Min. Eng.*, Sept. 1896.

minutes to 10, and in half an hour to 4 divisions,—showing that the current was reduced to one-fifth, and showing also that the rate of dissolution of the gold was reduced in like proportion. If, now, a porous partition *C* be placed in the vessel in order to prevent the solutions surrounding the different electrodes from mixing, and if air or oxygen be blown into the solution at the gold surface *B*, little or no change takes place in the current; but if the air or oxygen be applied at the pyrites surface *A*, a sudden increase in the current takes place, and the needle returns to its original position of 20 divisions or thereabouts,—showing that the gold is again dissolving at the same rate as when contact was first made. From this simple experiment it may be surmised that oxygen has no direct action in the dissolution of the gold. From this and other experiments which we need not go into here, we have concluded that at ordinary temperatures and pressures oxygen plays a secondary part only, in the dissolving of the metals.

What really happens is, that through the dissolving of the gold the current set up deposits hydrogen on the pyrites cathode, as already explained, which is then said to be *polarised*. This hydrogen has a solution pressure of its own, and opposes the solution pressure of the gold, which results in a falling off in the current strength as the hydrogen accumulates and becomes more concentrated. The effect of the air is to saturate the solution surrounding the pyrites with oxygen molecules, which induces the occluded hydrogen to become ionised and enter into combination, forming (1) hydrogen peroxide and (2) water thus—



A third reaction occurs between the oxygen and hydrogen ions to form water direct thus—



So long as the flow of air bubbles is kept up, so long will the current remain constant and the gold continue to dissolve at a high and uniform rate. As the area of the gold surface diminishes so also will the current diminish, and in like proportion the weight of metal dissolved in a given time.

The opposing solution pressure of the hydrogen varies with the amount of hydrogen occluded by the pyrites. Thus, when the hydrogen increases in density until its solution pressure is equal to that of the gold, dissolution must cease. In practice, however, this never occurs, for there is always a leakage of current, and consequently metal dissolved, due, we believe, to the oxygen dissolved in the solution continually combining with some occluded hydrogen. The affinity of hydrogen for oxygen is greater than that of gold for cyanogen, and it follows, if oxygen is available to oxidise the deposited hydrogen, dissolution of the gold can go on, but if no oxygen is available, the opposing forces soon balance each other and dissolution of the metal must cease.

The reason why the air, when applied to the surface of the gold, does not increase the dissolving action, is simply that, at that electrode it has no function to perform.

Oxidising agents have the same effect as air or oxygen gas to a greater or less extent, but the chemical changes that they bring about vary with the oxidiser. They all, however, accomplish the same end, which is to depolarise the negative electrode by combining with hydrogen as deposited.

It is thought by some chemists that base metals and minerals dissolve in the absence of oxygen, but that gold and silver dissolve only in its presence. As a matter of fact, all difficultly soluble metals, and for the most part metallic minerals, only dissolve to an appreciable extent, under ordinary atmospheric changes, in the presence of oxygen or an oxidising agent; the reason being chiefly that their solution pressure is too small to overcome the opposing electromotive force due to deposited hydrogen at the cathode, whereas the more oxidisable metals produce sufficient energy to overcome this opposing electromotive force, and can therefore proceed to dissolve even in absence of oxygen. The presence of oxygen, however, indirectly makes all metals more soluble.

The action of oxygen may be better understood from the following experiments:—Oxygen and hydrogen gases when mixed in a flask may be kept for an infinitely long time without chemically combining, but if a piece of moist platinised platinum be introduced into the mixture, union takes place and some water is formed. A similar action occurs if one piece of platinised platinum is charged with H and a second piece with O, and then short-circuited by a wire and immersed in water. A current is set up, flowing from the O electrode through the wire to the H, and thence through the water, the result being a union of the H and O, forming water. When the operation is conducted in presence of air, traces of  $\text{H}_2\text{O}_2$  are also formed, and if air be blown through the solution, the amount of  $\text{H}_2\text{O}_2$  formed is much increased. Again, if the two plates are connected with a primary cell and air is blown through the solution at the cathode, a considerable quantity of  $\text{H}_2\text{O}_2$  may be detected; and finally, if the source of the current be a dissolving metal in the solution and air be blown into the solution at the cathode, the same action takes place, i.e. the formation of  $\text{H}_2\text{O}_2$ .

Now, when  $\text{H}_2\text{O}_2$  is in contact with a cathode charged with H, one atom of the O combines with two atoms of the occluded H to form two molecules of water.

When dissolving gold in a clean cyanide solution in contact with pyrites, we have always been able to detect  $\text{H}_2\text{O}_2$  in the solution, particularly in that part surrounding the pyrites, but the quantity formed varies widely with the impurities in the solution. Bodlaender\* dissolved gold rapidly in an aerated solution, and not only detected the presence of  $\text{H}_2\text{O}_2$ , but also estimated a large quantity of that produced.

The action may be explained by the H occluded at the electrode exerting

\* *Zeit. angew. Ch.*, p. 583, 1896.



its solution pressure, which in the presence of O molecules is induced to break down into ions, two H ions combining with one O molecule. When, however, the conditions admit of O ions also being formed, as in the case above cited, or when a metal capable of occluding both gases is immersed in water, and a mixture of oxygen and hydrogen are caused to bubble through the water in contact with the metallic surface, the reaction becomes ionic and water is formed with only traces of  $H_2O_2$ .

The action of the air being entirely local, it cannot interfere with the current between the gold and pyrites electrodes, but as the hydrogen becomes ionised and oxidised, the current increases, by reason of removing the opposing electromotive force. In dry gases no such actions can take place, as the gases cannot become ionised, and therefore cannot combine.

The action of air or oxygen may be retarded, or entirely stopped, by the presence of reducing agents in the solution.

## CHAPTER X.

### DISSOLUTION OF THE GOLD AND SILVER.

#### Section II.

**The Effect of Reducing Agents.**—Potassium cyanide and alkali of commercial quality usually contain sulphides that act as deoxidising agents when in solution. J. Lœvy\* made some determinations of  $K_2S + Na_2S$  in good commercial cyanide, and found that it contained only 0·37 per cent., and that gold leaf dissolved just as quickly in a 0·5 per cent. solution of this salt as in a similar strength cyanide free from sulphides. He is of opinion, however, that higher percentages of sulphides may alter matters materially. A. F. Crosset† says,—“The conclusion I am forced to arrive at is that alkaline sulphides are not injurious when present only in very small quantities, if proper regard is had in the use of the cyanide solutions to the aeration of the solution; but a very small quantity of such sulphides may be injurious under conditions unfavourable to aeration of the cyanide solutions.” This is certainly true, but if the solution is properly aerated, sulphides can no longer exist.

Sulphides are sometimes introduced into the solution with alkali added to protect the cyanide. Good commercial caustic soda often contains a small percentage of  $Na_2S$  and ferro-sodium sulphide.

We have made some investigations on the effect of small quantities of  $Na_2S$  added to a 0·25 per cent. cyanide solution when used for dissolving gold and silver. Table XVI. gives the results. The weight of gold dissolved in a clean

TABLE XVI.

Na <sub>2</sub> S added per ton Solution in grms.	Relative Dissolving Power.	
	Gold.	Silver.
0	100	55
5	66	42·6
20	57·7	39·6
40	53·4	39
80	52	39
280	50·6	39

solution is taken as 100. That is its dissolving power, and the other figures represent the dissolving powers in terms of that solution.

\* *Proceed. Chem. and Met. Soc. S. Africa*, vol. i. p. 87.      † *Ibid.*, vol. i. p. 89.

These results have been plotted in fig. 16, and the two curves placed beside each other for comparison. It will be interesting to note that the

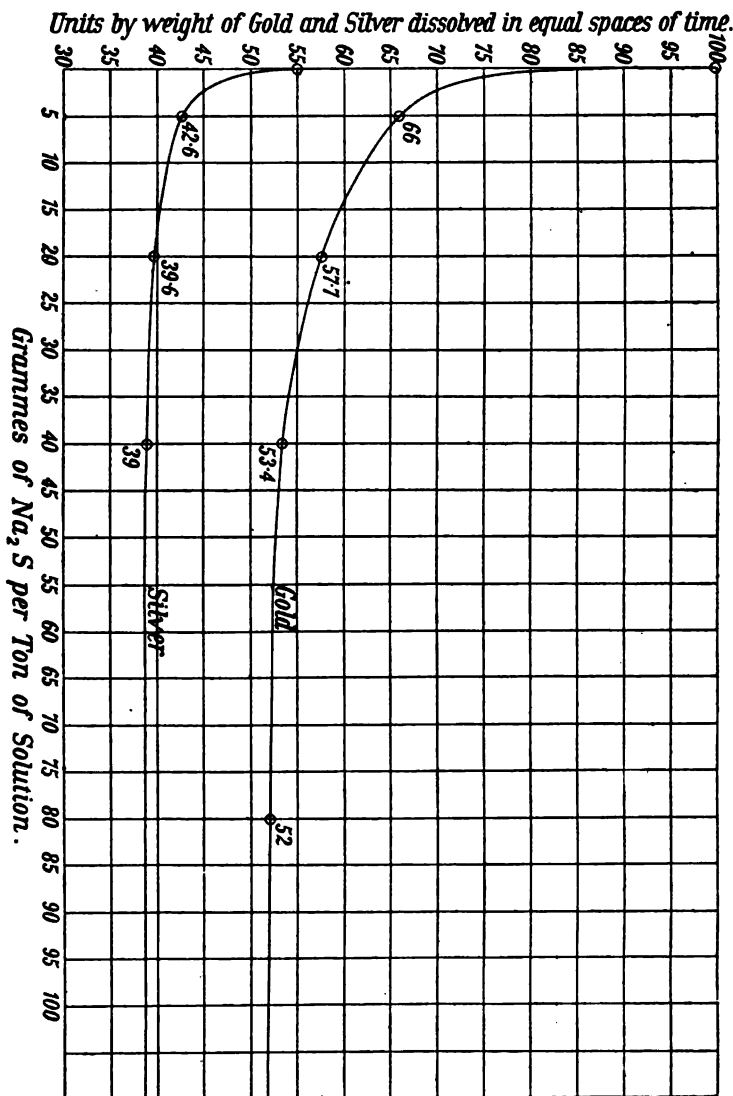


FIG. 16.—The effect of Sodium Sulphide in a 0.25 per cent. KCy Solution on the rate of dissolution of Gold and Silver.

solubility of silver is not affected to anything like the same extent as gold. In weaker cyanide solutions the silver becomes more affected and the gold less.

Many cyanide men have experienced that when a new solution has passed through an ore containing sulphides, tellurides, and other minerals, it is no

longer capable of dissolving the same quantity of gold in the same time as the original solution, although containing the same percentage of cyanide.

In order to prove that this action is real and not imaginary, and may be traced to some change in the solution, the following experiments were made. A well mixed sample of concentrates from the Main Reef series, Witwatersrand, containing chiefly iron pyrites (pyrite and marcasite), was divided into eight equal parts, and each part was placed in a separate treatment tank. These were then thoroughly washed, each in the same way, and a cyanide solution 0·3 per cent. equal in weight to the ore was slowly passed through No. 1 tank with a measured quantity of air. The filtrate was then passed on to No. 2 tank, and so on through all the eight tanks, each having been dealt with in identically the same way as No. 1. The results are given in Table XVII.

TABLE XVII.

Assay Value of Originals, 92 dwts. Gold.

No. of Tank.	Cyanide per cent.	Assay Value of Residues, dwts.	Gold per cent. extracted.
1	0·3	27·6	70
2	0·3	36·55	60·3
3	0·3	45	51·1
4	0·3	51·24	44·3
5	0·3	56·9	38·2
6	0·3	59	35·9
7	0·3	60·7	34
8	0·3	60·6	34·1

The solution had lost cyanide in passing through the tanks of ore, but was in each case made up to the original strength, 0·3 per cent., before passing on to the next tank. The solution was applied with a sprayer at a uniform rate, which gave it a better opportunity of taking up oxygen than would be obtained in practice. The time of treatment in all cases was the same, and, as far as practicable, identically the same method of procedure was maintained throughout. The residues were washed with a superabundance of air-free water before assaying. It is noteworthy, that in these experiments the volume of air passed through the ore with the solution has to be regulated and maintained alike in each case, or the results become erratic. The larger the volume of air used the worse is the resulting filtrate for re-use in dissolving gold and silver, but, as a rule, the better is the extraction of the metal from the ore treated.

It often happens that silver predominates in an ore, and it seemed probable that with such an ore, the above changes in the solution would have even a more marked effect. We have therefore treated a mispickel ore, in which silver predominates, identically in the same manner as the case above cited, except that the solution contained 0·4 per cent. cyanide. The results are given in Table XVIII.

The ore was one that was particularly suitable for treatment by the cyanide process, for by repeated treatments with clean cyanide solution 98·8 per cent. of the silver was extracted and about 80 per cent. of the gold.

That a change takes place in the solution in leaching the ore is beyond doubt, but what that change is requires investigation. After passing through eight tanks the solution in each case was of a dingy brown appearance. This changed to an amber tint after standing in contact with air and direct

TABLE XVIII.

Assay Value of Originals 18·3 dwts. Gold.  
 " " " 60·1 " Silver.

No. of Tank.	Cyanide per cent.	Silver in Residues, dwts.	Per cent. Silver extracted.	Gold in Residues, dwts.	Per cent. Gold extracted.
1	0·4	18	70	6·3	65
2	0·4	25·1	58·2	11·85	35·2
3	0·4	30·36	49·4	12·88	29·6
4	0·4	34·8	42·1	13·5	26·2
5	0·4	38·8	36·3	14·2	22·4
6	0·4	42	30	14·44	21·1
7	0·4	42·6	29·1	14·5	20·9
8	0·4	42	30	14·6	20·3

sunlight for about a week, but on adding an oxidising agent to a portion the change of colour was complete in a few minutes.

The solution in the first case, which had been a week in contact with air and sunlight, was then strengthened to 0·3 per cent. cyanide and passed through a ninth tank of the same ore in exactly the same way as the other eight solutions. The assays showed an extraction of 66·6 per cent. of gold as against 70 per cent. with the new solution. It would therefore appear that the solution had almost recovered its original dissolving power.

In the second case the solution was immediately treated with  $\text{Na}_2\text{O}_2$  and strengthened to 0·4 per cent., and within half an hour was applied to a ninth tank of ore, resulting in an extraction of 65·8 per cent. silver and 60·2 per cent. gold. From these experiments we may conjecture that reducing agents are accountable for the loss in dissolving power of the cyanide solution, and that this power may be regained by the addition of oxygen or oxidising agents.

**Removal of Reducing Agents.**—It is supposed that the reducing agents present which cause loss in dissolving power of the solution are alkaline sulphides, but it is probable that other reducing agents are also accountable for the action. If sodium sulphide is dissolved in water it slowly changes, in contact with air, to caustic soda and thiosulphate of sodium. The necessary oxygen for this change must come from that dissolved in the solution. The oxidising of  $\text{Na}_2\text{S}$  into  $\text{Na}_2\text{S}_2\text{O}_3$  by this means is retarded by the presence of caustic soda in excess, but hastened in presence of a car-

bonate or  $\text{CO}_2$ . When an energetic oxidiser, as  $\text{O}_3$  or  $\text{H}_2\text{O}_2$ , is employed the same action takes place, but the oxidation is more complete, forming finally  $\text{Na}_2\text{SO}_4$ .

If cyanides are present in the above solution a small portion also becomes oxidised by the action of the air, and with more energetic oxidisers the amount may be much increased. With weak solutions the loss from this source is, however, not sufficient to be a matter of importance. A little of the sulphur combines with the cyanide to form thiocyanate, but this compound does not appear to have any retarding influence on the dissolving of gold and silver.

The oxidising of sulphides introduces into the solution compounds that may not have any marked influence on the dissolving power of the cyanide, but the increase of salts in the solution, even inert salts, is generally undesirable. Attempts have been made to remove the sulphur as insoluble compounds, and in March 1893 J. S. Macarthur and C. J. Ellis patented the addition of a lead salt to the solution, which had for its object the removal of sulphur as insoluble lead sulphide. Other desulphurising agents were also suggested. This method will remove excess of sulphur, but its action, when only traces of sulphides are present, as is usually the case, is not very effective. Even after the most complete precipitation of sulphur, the solution should be thoroughly aerated or treated with an oxidiser before it is used.

**The Application of Oxidisers.**—In 1890–91 Julian made a large number of experiments on the use of air blown through working solutions, and on the addition of more energetic oxidising agents, for the purpose of facilitating the dissolution of gold in ores. The results were so promising that in February 1892, he covered the use of air blown through the solution, and of the use of the more energetic oxidising agents for this purpose, by Transvaal patent No. 335. About the same time C. Moldenhauer, of Frankfort-on-Maine, was experimenting in the same direction, and patented in the United States in the same month and the same year the use of ferricyanide of potassium as an oxidiser in conjunction with potassium cyanide, to hasten the dissolving action on gold and silver. It was supposed at that time that the only use of oxidisers was merely to supply the oxygen direct for the so-called Elsner reaction instead of taking it from the air; but we now know that the direct use of oxidisers for dissolving purposes has a very limited application, and that their chief value lies in oxidising reducing agents in the solution.

Julian experimented with a large number of organic and inorganic oxidisers, and it was found that many are undesirable, owing to their forming soluble compounds that increase the density of the solutions, and in some cases forming complexes with the cyanide of no value in the process. This increase of density has generally the effect of decreasing the solubility of the metal, by offering greater resistance to the diffusion of the dissolved oxygen molecules. It is therefore advisable, when employing oxidisers, to select one that introduces only oxygen or oxygen and water. The more energetic

oxidisers, as  $\text{KMnO}_4$ ,  $\text{Na}_2\text{O}_2$ ,  $\text{O}_3$ , etc., oxidise cyanide to cyanate, and facilitate the formation of some complex compounds, but in very dilute solutions this action is not marked.

The solution should be charged with oxygen before being applied to the ore, and reducers oxidised as completely as time will admit. This may be done by blowing in air vigorously for a few minutes at intervals of an hour, but the action is slow, and in cold weather it may take some days, in this way, to render reducers inert, and saturate the solution contained in large tanks. If the operation is kept up continuously a considerable volume of  $\text{HCy}$  may be carried off, besides being a source of unnecessary expense. Another method is to pump the solution from one tank to another, employing a pump with a leaky suction pipe. Air is then drawn in with the solution and a thorough mixture ensured, especially if the pump is one of high velocity, like those of the centrifugal type. The selection of methods must, however, be largely governed by cost. Both the above methods have the advantage of thoroughly mixing the solution and making it of uniform strength.

Probably at no works is sufficient time allowed to completely oxidise all reducing agents and re-charge the solution with oxygen by the above methods alone, and from diagram fig. 16 it will be seen how injurious is even a mere trace of sodium sulphide. Where time of treatment is not a matter of importance, the oxidation of sulphides is not so necessary, for if the dissolving power of the solution is reduced by their presence, we have only to increase the time of treatment in the inverse proportion. It is noteworthy that in works where the sump capacity is small, high residues much more often occur than in works where the capacity is large. This may be accounted for by the solution being longer in contact with the air before it is applied to the ore, thereby preventing the accumulation of reducers.

Oxidisers are sometimes added as solution to the ore before applying the cyanide solution. This is found beneficial when the ore has become partly oxidised, and in the case of slime treatment by agitation. Some of the more easily soluble mineral compounds are then made less soluble in the cyanide solution.

Strong oxidisers, as originally patented by Julian and Moldenhauer, have been applied to the treatment of tailings and concentrates in many parts of the world, and for the most part, with the former material, have not proved economically beneficial. In 1894 Sulman and Teed brought to the notice of cyanide men the use of a halogen compound of cyanogen, as  $\text{BrCy}$ , in a cyanide solution for hastening the dissolution of gold and silver. The compound is now in use at several mines, particularly in W. Australia. There has been much discussion as to the action of  $\text{BrCy}$ , and we have made some investigation on the subject. We find that its only action of value is that of an oxidising agent. It has been stated by Sulman and others that it liberates cyanogen. This it probably does, like other oxidising agents, but, as pointed out by W. Skey,\* "aqueous solutions of cyanogen do not

\* *Eng. and Min. Jour.*, page 163, 1897.

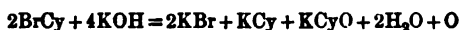
exert the least solvent action on gold and silver." The liberating of cyanogen is really a weak point in the use of strong oxidisers, which will be seen from the reaction—



The reactions of value that occur are similar to other oxidising agents, such as  $\text{K}_3\text{FeCy}_6$ ,  $\text{KMnO}_4$ , etc., and are probably for the most part as follows:—



or with excess of  $\text{KOH}$ ,



with  $\text{KCy}$ ,



or with  $\text{KCy}$  and an excess of  $\text{KOH}$ ,



The effect of this and other oxidisers may be demonstrated by the use of apparatus fig. 15, when it will be found that no action takes place if the oxidiser is applied to the gold surface, but if applied to the pyrites surface a strong current is generated, and consequently a rapid dissolution of the gold must occur.

The value of strong oxidisers when applied to ores is determined largely by the velocity of the reaction. Those which have too high a velocity become wasted through the oxygen being liberated faster than it is wanted to combine with the depositing hydrogen. The liberated oxygen then destroys part of the cyanide. It is obvious that the ideal oxidiser is one by which the oxygen is liberated just at the rate required to combine with the deposited hydrogen. There must, however, always be a large excess of oxygen, as we can never ensure the liberation to take place at the right point, but must rely on diffusion.

It is obvious from the above conclusions that it would not be economical to apply strong oxidisers to an ore direct, as, owing to the rapidity of the reaction, the effect lasts too short a time to facilitate the dissolution of the gold to an appreciable extent. When the gold is coarse, it must have a considerable time allowed to dissolve, and any oxygen given off beyond that necessary to combine with the hydrogen generated by its dissolution cannot be of service, but may be a source of retarding dissolution, through destroying cyanide. The only case in which strong oxidisers can be of benefit, when applied to an ore direct, is where the gold is in an extremely fine state of division and can dissolve rapidly, as in the treatment of slimes by agitation, a case in point being the telluride ores of W. Australia.

If an oxidising agent is applied to an ore dealt with by percolation, one should be selected that gives off oxygen slowly, as  $\text{MnO}_2$ ,  $\text{BaO}_2$ ,  $\text{PbO}_2$ . These all act as depolarisers, the former being the slowest and the latter the most rapid. The peroxides of the alkalis or combinations of  $\text{H}_2\text{O}_2$  are better employed in oxidising reducing agents in the solution before applying to the ore.



In some ores the gold is so easily dissolved that no marked disadvantage is observed in using solutions of low dissolving efficiency, for the reason, as pointed out by C. Butters and W. Bettel,\* that it takes the best part of three days to wash out the gold, after it is dissolved, from a tank of ore, and as the washing out is done with a cyanide solution, the dissolving of the gold proceeds at the same time. Hence if a solution were used of such a high efficiency that the whole of the gold was dissolved in one day, a solution of only half that efficiency would theoretically answer as well if the washing out of the dissolved gold took two days. In practice, however, by using clean solutions of high efficiency, we can rely on uniformly good results, whereas with solutions of low efficiency, the results are usually erratic, even with the simplest class of ore.

When dealing with refractory ores, and especially those containing much silver, it often takes a long time (sometimes three or four weeks) to dissolve the gold, whereas, when the solution is kept up to its highest state of efficiency, the time required may be less than half, and the results more uniformly good.

\* *Proceed. Chem. and Met. Soc. S. Africa*, vol. i. pp. 331-332.

## CHAPTER XI.

### DISSOLUTION OF THE GOLD AND SILVER.

#### *Section III.*

**Strong and Weak Solutions.**—The solubility of gold and silver in cyanide solutions is dependent, at ordinary temperatures and atmospheric pressures, on two main factors: (1) the number and velocities of the migratory ions and oxygen molecules that take part in the dissolving of the metals; (2) the difference of potential between two parts of the metallic surface in contact with the solution. Now, within working limits, the stronger the solution the greater is the number of dissociated molecules or ions in it, but at the same time the more viscous does it become. Increase in viscosity means greater resistance to the movement of the ions and molecules, and therefore the stronger the solution the slower will be their movement. For this reason no proportionality can exist between the strength of the solution and its dissolving power. This will be better understood by reference to fig. 17. Let the vertical axis on  $OA$  represent equal unit weights of metal dissolved in equal spaces of time, and the horizontal axis on  $OB$  represent the percentage strength of the cyanide solution. Then if the weights of metal dissolved were in proportion to the strength of the solution, the straight line  $OC$  would represent the weights of metal dissolved by the solutions of different strengths, but from a set of experiments the curve  $OD$  was found to represent the weights of metal dissolved. For example, if the weights dissolved were in proportion to the strengths of the solutions,  $b\ c$ , which is 50 units, would represent the weight of metal dissolved in a 0.5 per cent. solution, but it was found that the actual weight dissolved was only about 25 units, equal to  $b\ d$ . Therefore  $d\ c$ , equal to 25 units, shows the decrease in the dissolving power, due chiefly to the increased viscosity of the solution. As might be expected, the more dilute the solution the nearer does the dissolving power approximate to proportionality. Thus it will be seen that for solutions more dilute than 0.1 per cent., the curve  $OC$  nearly follows  $OD$ .

Maclaurin \* made some interesting experiments to show how the viscosity of a cyanide solution affects the solubility of gold and silver. He found that the amount of metal dissolved in strong solutions was less than the quantity of oxygen absorbed appears to demand, and he thinks that this points to a retarding action by the salt molecules to the motion of the oxygen molecules.

\* *Jour. Chem. Soc.*, vols. lxvii., lxviii. p. 208.

The potential difference between the gold and the electro-negative surface varies with the latter, although the chemical changes are the same in the same solution. Thus the potential difference between gold and iron pyrites in a cyanide solution is different from the potential difference between gold and galena, although in both cases the gold dissolves, forming  $\text{KAuCy}_2$ , the difference being that the one of the greater potential difference dissolves the faster.

**Polarisation.**—In the above we have assumed that there is no polarisation, or rather that the hydrogen is oxidised as fast as it is deposited at the negative electrode. In practice, however, polarisation always takes place, and we have to deal with matter from a practical point of view. The time of polarisation varies with the relative sizes of the electrodes, that is, with

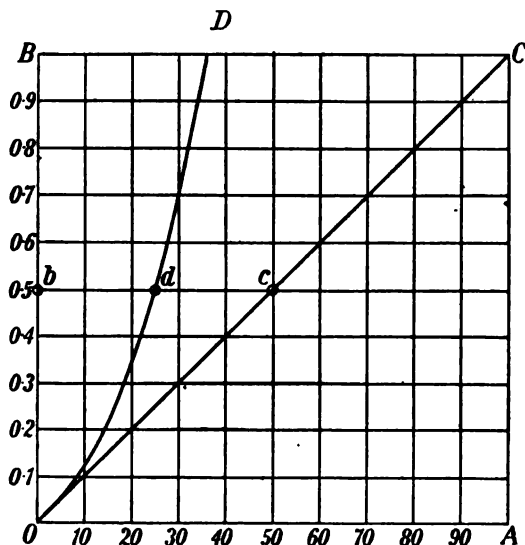


FIG. 17.—Ratio of Weight of Metal dissolved to strength of Solvent.

the relative surface areas of the gold and pyrites or other negative substance in contact with the solution, also it varies with the percentage of oxygen absorbed by the solution, and with the resistance offered to the motion of the oxygen molecules in the solution. As we are not capable of determining the values of all these factors, and also as they are constantly varying during the dissolution of the metal, it is not possible to obtain data that represent the loss in solubility of the metal due to polarisation, when treating ores with solutions of different strengths.

What we know, however, is that the stronger the solution the quicker does the electrode polarise, and the greater the negative or pyrites surface, or the smaller the gold surface, the slower does polarisation take place. Also, the stronger the solution the greater is its viscosity, and the more viscous the solution the greater is the resistance offered to the diffusion of the oxygen molecules through it, therefore the slower is depolarisation.

It is evident there must be a solution of a maximum strength with a minimum resistance to the oxygen molecules. This would have a maximum dissolving power, but the strength must necessarily vary with different sets of conditions. In practice no set of conditions remains constant, but changes during the dissolving of the gold, and changes with ores from different mines, or even the same mine. It is therefore possible only to find a solution that is a rough approximation to one of maximum dissolving power. This can so far be found only in an empirical way, and must obviously vary widely in different localities.

**Physical effect of Strong and Weak Solutions.**—In practical work the strength of the solution was found, in some cases, to have a marked physical effect on the dissolving of gold and silver. This is most noticeable in ores containing a preponderance of silver over gold in presence of sulphides. Thus it was found that when employing a weak solution of 0.15 per cent. KCy a maximum extraction point of 66.6 per cent. silver was reached, and even when the solution was afterwards strengthened to 0.5 per cent. the additional extraction was very small. When, however, the first solution applied was 0.5 per cent., and afterwards weaker solutions were used, the extractions were always over 90 per cent.

It is surmised that the weak solution produced a hard insoluble film of sulphide over the surface of a portion of the metallic silver particles, which prevented even the strong solution from attacking the silver, whereas, when the strong solution was used at first, the sulphide adhered as a loose slimy deposit, which did not prevent the weaker solutions afterwards attacking the metallic silver. A hot solution produces the same slimy deposit, even when the solution is weak.

**Comparative Solubility of Metals and Minerals in Cyanide Solutions.**—J. S. Maclaurin \* made some interesting investigations on the solubility of gold and silver in strong and weak cyanide solutions. He dealt with plates of pure gold and silver, and found that the rate of dissolution of both metals varies with the strength of the solution, being small for strong solutions, and increasing as the solution becomes weaker until a maximum at 0.25 per cent. KCy is reached, and then again diminishing.

Maclaurin used pure gold plates, in which case the difference of potential between the dissolving part or anode, and the polarisable part or cathode, is a minimum. Under such circumstances the rate of dissolution of the gold must be small, as its electro-motive force is soon balanced by the opposing electro-motive force of the deposited hydrogen, and all action would cease unless oxygen molecules struck the polarised part and effected oxidation of the hydrogen. In dealing with a pure gold plate in a cyanide solution, we have found that the maximum difference of potential, at any two parts of its surface, is  $\frac{1}{100}$  to  $\frac{1}{100}$  of the difference of potential between gold and marcasite. This accounts for pure metal being much less soluble than gold in ores.

\* *Jour. Chem. Soc.*, vols. lxvii., lxviii. pp. 199-212.

In ores, we have never such a set of conditions as those in Maclaurin's investigation, for we usually find a small speck (often microscopic) of gold and silver in contact with a comparatively large negative area of pyrites or other mineral. In this case we have a much greater difference of potential, which causes the rate of dissolution to be proportionately greater. Complete polarisation never occurs while oxygen is present in the solution, for when the negative area is large compared with the gold, it polarises more slowly, and favours depolarisation by meeting with a larger number of oxygen molecules than if the reverse was the case. Also the gold and silver particles are often so small, compared with the pyrites surface, that the metals dissolve completely before polarisation has any appreciable effect. We have found that with positive and negative electrodes of the same size, the time of practically complete polarisation between two pure gold electrodes was under one minute, whereas, under the same conditions, where the electrodes were gold and iron pyrites, polarisation was not complete in half an hour.

In dissolving pure gold and silver plates in a cyanide solution, the amount of oxygen dissolved has a very sensible effect on the solubility of the metals, but in the case of ores, where large negative and small positive surfaces present themselves to the solution, the effect of oxygen is not so marked, and in fact varies but slightly within comparatively wide limits. The large excess of oxygen becomes, in the latter case, used up in oxidising impurities dissolved in the solution, such as sulphides. We therefore do not get the full benefit of the oxygen for depolarising purposes.

Much confusion exists in the minds of cyanide men and others as to the meaning of the expression 'comparative solubility of metals.' We treat it as meaning the comparative weights of metal dissolved from surfaces of equal area in the same time and under the same set of conditions. It is often confounded with the energy with which a metal is attacked by a solution, whereas in reality there is no proportionality between the two. Thus it is often said, because of the energy displayed in dissolving, that aluminium in a cyanide solution is more soluble than zinc, and zinc is more soluble than copper, but from experiment we find that, under similar conditions, the weights of these metals dissolved are in the order 1, 3, and 2 respectively, while the ratio of the energy produced is about 100, 93, and 84 respectively.

It was thought by some metallurgists that the proportion of gold and mineral matter of the ore dissolved in a weak solution, differed from the proportion dissolved in a strong solution. That is to say, that the ratio of gold to pyrites dissolved in a weak solution differs from the ratio of that in a strong solution. We have made some investigations on the subject, and find that this is not the case, but that the same proportionality exists with solutions of all strengths within working limits. The data obtained are given in Table XIX. The first vertical column gives the order of solubility, and the other vertical columns give the ratios of solubility in solutions of strengths indicated at the top of each. The horizontal rows

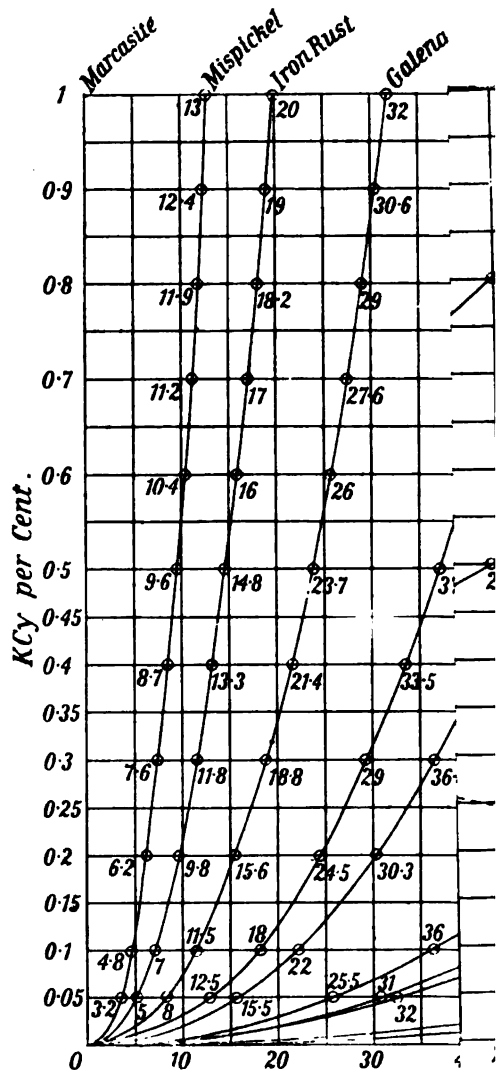
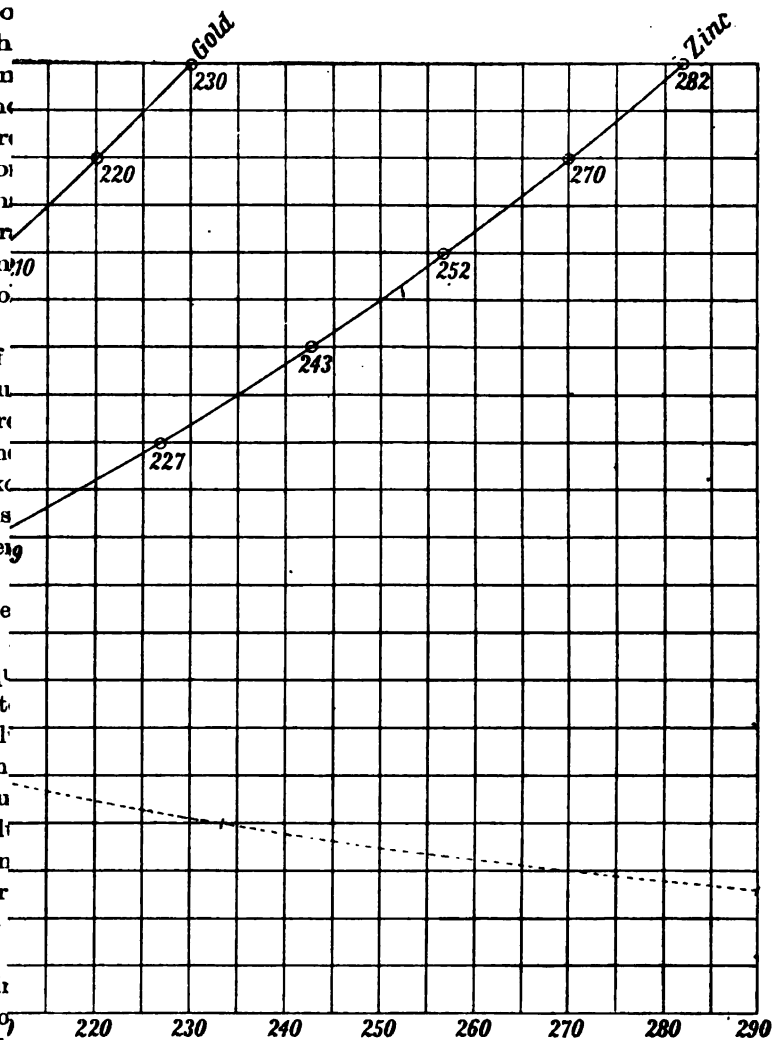


FIG. 18.—Th<sup>liss</sup>

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ing efficiency of KCy per unit of 0.01 per cent.

of figures indicate the ratios of the weights of metal or mineral dissolved in solutions of the different strengths, from surfaces of equal area in equal spaces of time. These results are shown graphically in fig. 18.

As might be expected, the results obtained with the minerals varied widely

TABLE XIX.

Order of Solubilities.	Per cent. KCy.										
	1	0.9	0.8	0.7	0.6	0.5	0.4	0.3	0.2	0.1	0.05
Zinc, . . . .	282	270	252	243	227	209	190	166	138	100	71
Gold, . . . .	230	220	210	198	185	170.5	154	135	112.5	82.5	58
Silver, . . . .	127	121.5	116	109.5	102	94	85	75	62	45.5	32
Lead, . . . .	121	116	110.5	104	97	90	81	71	59	43	31.3
Copper pyrites, . . . .	101	96.6	92	87	81	75	67.5	60	50	36	25.5
Charcoal iron, . . . .	62	59	56	53	50	46	41.5	36.5	30.3	22	15.5
Pyrite and cast iron, . . . .	50	48	45.5	43	40	37	33.5	29	24.5	18	12.5
Galena, . . . .	32	30.6	29	27.6	26	23.7	21.4	18.3	15.6	11.5	8
Iron rust, . . . .	20	19	18.2	17	16	14.3	13.3	11.3	9.3	7	5
Mispickel, . . . .	13	12.4	11.9	11.2	10.4	9.6	8.7	7.6	6.2	4.8	3.2
Marcasite, . . . .	0	0	0	0	0	0	0	0	0	0	0
Efficiency of KCy } per unit, . . . }	0.096	0.102	0.109	0.118	0.129	0.142	0.160	0.188	0.234	0.344	0.483

with samples from different localities, and therefore an average sample was dealt with. Of the more common metallic minerals, marcasite was found to be the least soluble in cyanide solutions. The solubility of the other substances is therefore given in terms of this mineral, which is regarded as insoluble in all the solutions. This assumption is, of course, not strictly accurate, but gives us a basis for comparison.

The figures were obtained under conditions that produce no polarisation. This is essential for purposes of comparison, as it would be difficult, if not impossible, to maintain the same degree of polarisation during each successive set of determinations. A large number of the values were obtained from actual experiment, and the others were determined by calculation.

TABLE XX.

	Per cent. KCy.							
	1	0.5	0.25	0.1	0.05	0.02	0.01	0.005
Gold, . . . .	0.00650	0.00670	0.00684	0.00675	0.00666	0.00613	0.00345	0.00030
Silver, . . . .	0.00395	0.00400	0.00410	0.00396	0.00380	0.00285	0.00213	0.00040

Maclaurin \* made a number of determinations of the solubility of gold and silver in solutions containing percentages of cyanide similar to the above. Table XX. are his figures, arranged for comparison.

\* *Jour. Chem. Soc.*, vols. lxvii., lxviii. pp. 199-212.



By Maclaurin's method of making these determinations it would have been impossible to prevent polarisation, although he took the precaution of giving motion to the plates, and of keeping the solution charged with oxygen. In dealing with pure gold or silver plates in a cyanide solution, without being in contact with anything more electro-negative, the difference of potential between any two parts of their surface is so small that an infinitesimal amount of hydrogen deposited on a part of its surface reduces the solubility in a marked degree. It would be expected that this deposition of hydrogen would have less effect in a weak solution, as the oxygen meets with less resistance, and therefore diffuses more readily to the part where the hydrogen is deposited, and effects depolarisation. The solubility in the weak solution should therefore approximate to that when no polarisation occurs,

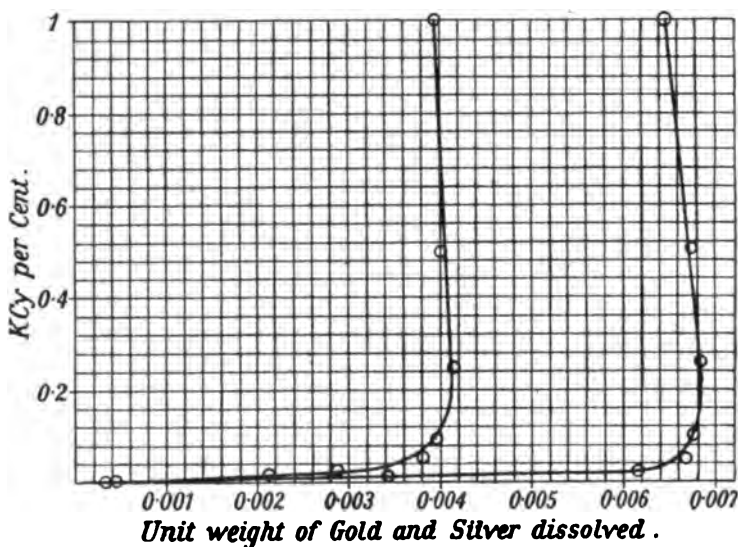


FIG. 19.—Maclaurin's Curves, showing Solubility of Gold and Silver plates in solutions ranging from 0 to 1 per cent. KCy.

but as the solution becomes stronger the resistance to the movement of the oxygen molecules increases, and consequently the solubility of the metal decreases. This is really what occurs, as shown by Maclaurin's results, which we reproduce graphically in fig. 19.\*

We are not able to place these curves beside the curves in fig. 18 for want of data to determine the scale, but we give them to show the general direction the solubility of the metals takes in an extreme case, that is, when the gold and silver occur in ores without being in electrical contact with a more negative substance. When the gold and silver are in very fine states of division and in electrical contact with a comparatively large negative area, as pyrites, the general direction of solubility approximates to that given in fig. 18.

\* *Jour. Chem. Soc.*, vols. lxvii., lxviii. p. 206.



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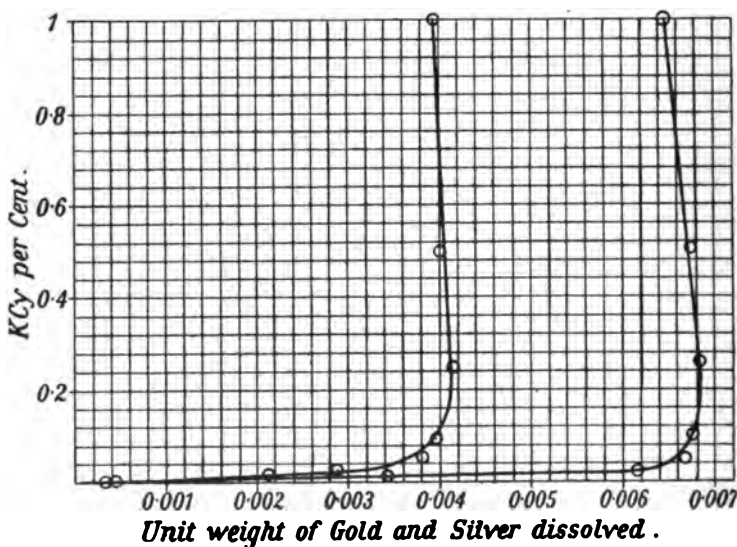


FIG. 19.—Maclaurin's Curves, showing Solubility of Gold and Silver plates in solutions ranging from 0 to 1 per cent. KCy.

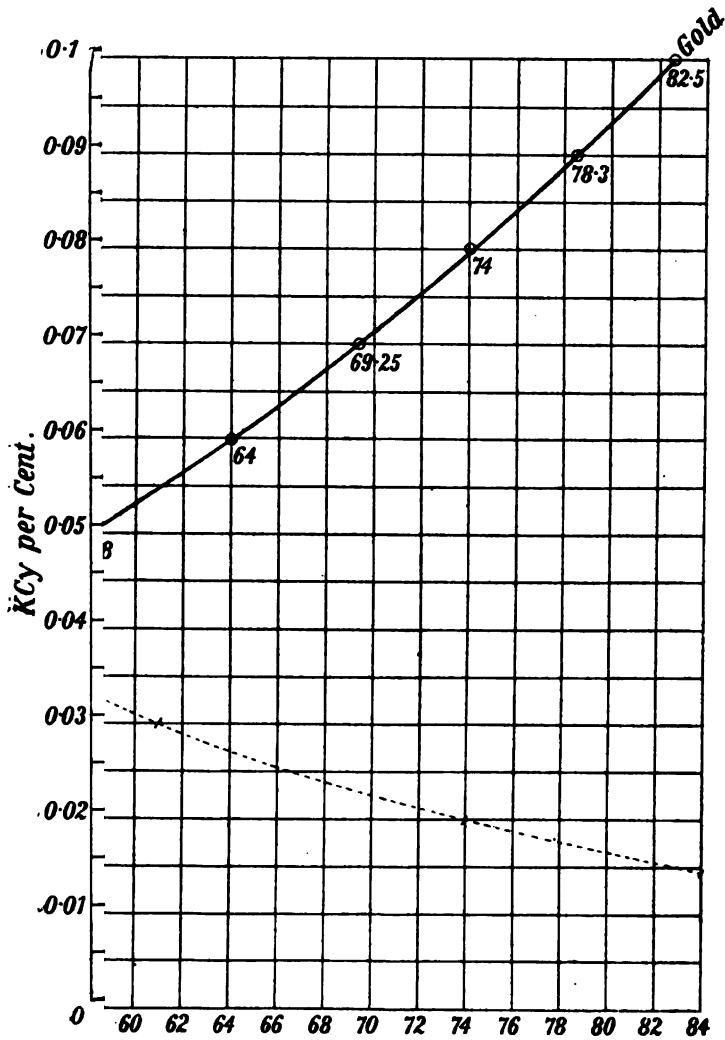
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\* *Jour. Chem. Soc.*, vols. lxxvii., lxxviii. p. 206.



[To face page 88.]



solving efficiency of KCy per unit of 0.1 per cent.

We have made a number of exact determinations of the solubility of gold and silver in very dilute solutions, in a similar manner to those given in Table XIX. The results are given in Table XXI., and are shown graphically in fig. 20 on an enlarged scale.

**Efficiency of Strong and Weak Solutions.**—On studying Tables XIX. and XXI. it will be seen that there is no proportionality between the weight of metal dissolved and the percentage strength of the solution. Thus a 0·01 per cent. solution dissolves 24 of gold, whereas a 0·1 per cent. solution, ten times as strong, dissolves only 82·5 in the same time, and a 1 per cent., or one hundred times as strong, dissolves only 230 parts instead of 2400, as would be the case if proportional.

Therefore the weaker the solution the greater is its efficiency per molecule of KCy dissolved, but in practical work this rule can easily be carried too far. A solution that would have a maximum dissolving power and a maximum efficiency per molecule of KCy is what we desire for working strength. This can, however, be arrived at only in an empirical way, as the factor time has to be taken into account.\* Let us assume 0·01 per cent. as a minimum working strength and call this the *working unit*; we may then

TABLE XXI.

Order of Solubilities.	Per cent. KCy.									
	0·1	0·09	0·08	0·07	0·06	0·05	0·04	0·03	0·02	0·01
Gold, . . .	82·5	78·5	74	69·25	64	58	51·5	44·5	35·5	24
Silver, . . .	45·5	43·3	40·8	38·2	35·3	32	28·4	24·6	19·5	13·2
Efficiency of KCy per unit }	0·344	0·363	0·385	0·412	0·444	0·483	0·536	0·618	0·740	1

easily obtain the dissolving power per working unit of KCy, by simply dividing the given ratios of the weights of metal dissolved by the number of working units in the strength of the solution. Thus, in the case of gold, Table XXI., the weight of metal dissolved in a solution when the strength is unity (0·01) is 24, and we may call the dissolving efficiency of this solution 24. In a solution of 0·1 per cent., which contains ten working units, we get  $82·5 \div 10 = 8·25$  per unit; and in a 1 per cent. solution, which has one hundred working units, we get 2·3 per unit. That is to say, that the dissolving efficiency of 0·01, 0·1, and 1 per cent. solutions is 24, 8·25, and 2·3 per unit of KCy respectively.

\* If the factor time is left out of the question, as in cases where the gold and silver dissolve rapidly, but can be washed out of the ore only slowly, the most efficient strength of solution is then from 0·07 to 0·09 per cent. KCy.

We have calculated the dissolving efficiency of KCy per unit for strengths of solution between 0.01 and 1 per cent. in terms of the working unit which we assume to have a dissolving efficiency of 1, and for convenience have placed these at the foot of the columns in Tables XIX. and XXI. We have also plotted the results multiplied by 1000 in fig. 18 and by 100 in fig. 20. Now, if we know the time required to treat an ore with a solution of known strength, and it is desired to vary the capacity of the works by lengthening or shortening the time of treatment, we may then determine from the data given in the tables, what strength of solution would give equally good results. Thus, if it required 200 hours to treat an ore effectively with a 0.01 per cent. solution, and it became necessary to increase the capacity of the works by decreasing the time of treatment to 70 hours without decreasing the extraction, what we have to do is to find in the columns a dissolving efficiency unit which multiplied by 200 equals 70 or thereabouts. Then the strength of the solution at the head of this column is that required. In this case it will be found  $0.344 \times 200 = 68.8$ . The required strength of the solution is therefore that at the head of the column above 0.344, which is 0.1 per cent. If 100 hours could be allowed, then the solution of 0.05 per cent. would answer equally well.

The results are not quite accurate, as polarisation is not allowed for, but for all practical purposes, with solutions up to 0.5 per cent. they are sufficiently near.

On examining the tables and curves, one of the reasons why, in practice, a weak solution is preferred to a strong one presents itself very forcibly, viz., the great loss in dissolving efficiency with increase in strength of solution.

## CHAPTER XII.

### TEMPERATURE EFFECTS.

**The Influence of Temperature on Dissolution of Metals.**—So far we have omitted the influence of temperature, but it is well known that this factor has a very marked effect on the rate of dissolution of metals.

H. C. Jones and J. M. Douglas\* have proved that the amount of ionisation in solutions of salts is not affected by temperature. Now, it seems beyond doubt that the solubility of a metal is dependent largely on the amount of ionisation of the salt or acid, and on the velocity of the ions. And as we know that the rate of dissolution of a metal is increased with rise of temperature, and as this cannot be due to greater ionisation, we may safely conclude that the increase in dissolution is due largely to an increase in the velocity with which the ions move. There are, however, other reasons for the increase of dissolution, as we shall see later.

We have made a number of determinations of the solubility of gold in a 0.25 per cent. KCy solution, at temperatures between those of the freezing and boiling points of water. The results are given in Table XXII. and are plotted in fig. 21. The first and second columns give the temperatures in degrees Centigrade and Fahrenheit. The third column gives the unit weights of gold dissolved in equal spaces of time when no polarisation occurs. It will be noticed that at 0° C. 90 units were dissolved, and that with increase of temperature the solubility diminishes to 89 at 1 to 2 degrees, and then there is a gradual increase in the rate of dissolution until a temperature of 85° C. (185° F.) is reached, when the rate becomes 151. At higher temperatures there is a gradual decrease, until at boiling point it fell to 146.

With rise of temperature there must be a decrease in the quantity of oxygen absorbed by the solution; nevertheless we get no corresponding diminution in the rate of dissolution, but, on the contrary, the rate steadily increases. We were unable to obtain satisfactory data of the quantities of oxygen dissolved in a cyanide solution at different temperatures, but in absence of these, and to meet our purpose, we have taken the data given by Winkler† of oxygen dissolved in water, and assume that in a cyanide solution the amount dissolved would not differ greatly from these. We have multiplied Winkler's results by 1000, placed the products opposite the different temperatures and plotted them in fig. 21. By this means we are readily able to compare the rate of dissolution of the gold with the relative

\* *Am. Chem. Jour.*, vol. xxvi. No. 5, Nov. 1901.

† *Berichte d. d. chem. Gesell.*, xxiv. 3609.



amount of oxygen absorbed at the various temperatures. Now, it will be observed on examining the table or the curves that the solubility of oxygen decreases with rise of temperature, while the solubility of gold increases to a maximum at 85° C. and then slightly decreases to the boiling point, although

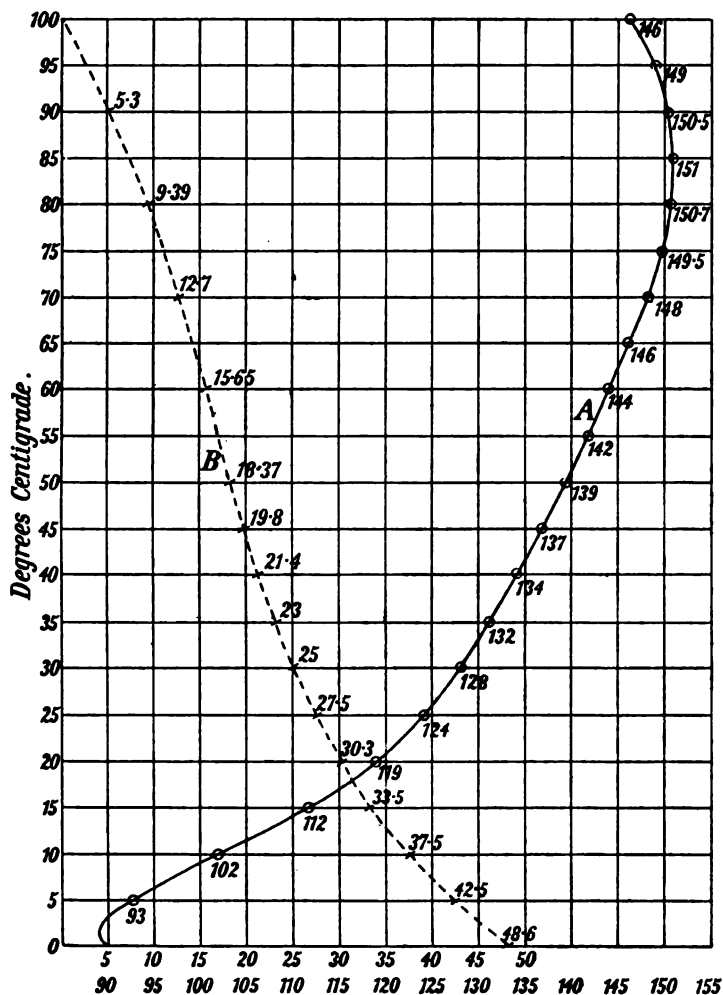


FIG. 21.—Temperature effects,—showing the Solubility of Gold in a 0.25 per cent. KCy solution at temperatures ranging from freezing to boiling points of water, and the relative Solubility of Oxygen in water at the same temperatures.

the oxygen solubility at that temperature is zero. It would appear from this that oxygen has no direct influence on the solubility of the gold. This is what we have already shown to be the case.

In seeking for the reason why oxygen should not exert its influence at the higher temperatures as it does at the lower temperatures, we have found

that the negative or pyrites electrode polarises to a less degree as the temperature rises. That is to say, that the capacity of the electrode to occlude or retain hydrogen at its surface is less in a heated solution than in the cold. This being the case, it follows that the maximum opposing electro-motive force due to polarisation becomes less and less as the solution becomes heated, until the E.M.F. of the dissolving gold overbalances this counter-force, and dissolution of the gold can then go on without the presence of oxygen. True, the E.M.F. of the gold also becomes less as the temperature

TABLE XXII.

Temperature.		Unit Weights of Gold dissolved.	Oxygen Solubility in Water $\times 1000$ .
C.°	F.°		
0	32	90	48.6
1	33.8	89	46
3	37.4	90	44.8
5	41	98	42.5
10	50	102	37.5
15	59	112	33.5
20	68	119	30.3
25	77	124	27.5
30	86	128	25
35	95	132	23
40	104	134	21.4
45	113	137	19.81
50	122	139	18.37
55	131	142	...
60	140	144	15.65
65	149	146	...
70	158	148	12.7
75	167	149.5	...
80	176	150.7	9.39
85	185	151	...
90	194	150.5	5.3
95	203	149	...
100	212	146	0

rises, but this effect is slight compared with the falling off in the opposing E.M.F. due to polarisation.

At the ordinary or low temperatures the balance of the two opposing forces is in favour of the hydrogen, and the deposit of this goes on increasing until its E.M.F. is in equilibrium with the E.M.F. of the gold, and then, in order that dissolution should continue, the presence of oxygen becomes necessary to oxidise the hydrogen. When the electrodes are in this state we get for each atom of hydrogen oxidised an atom of gold dissolved. If there is no oxygen present no hydrogen becomes oxidised, and the two E.M.F.s being in equilibrium, dissolution of the gold cannot proceed until the hydrogen electrode (pyrites) becomes of a lower potential. This may be done in two ways, either by continuously oxidising the hydrogen as it separates, or by

raising the temperature to such a point that the maximum E.M.F. of the hydrogen is less than the E.M.F. of the gold.

Referring again to Table XXII. and fig. 21, it may be noticed how the rate of dissolution of the gold increases with rise of temperature from about 3° or 4° C., at first rapidly, and then more gradually until a maximum is reached, and finally decreases. This maximum point was found for a 0.25 per cent. solution to be at 85° C. (185° F.), but it varies with solutions of other strengths and with impurities in the solution. Decomposition of KCy occurs even at ordinary temperatures, and the rate of decomposition goes on increasing with rise of temperature, but as the temperature rises, the velocities of the K and Cy ions increase, as already stated. Therefore, with increase of temperature we get an increase in the rate of dissolution of the gold, due to increase in the velocities of the K and Cy ions, but as the temperature rises a decomposition factor is introduced which reduces the number of K and Cy ions, and therefore causes a decrease in the rate of dissolution. Now, it may be conceived that the effect of the reduction in the number of these ions may more than counterbalance the effect of the increased velocity of those remaining, and thereby the net result must be a slower dissolution of the gold. This is evidently why a maximum is reached in the rate of dissolution before the temperature reaches boiling point. With a weaker solution this maximum is not reached so soon, but with stronger solutions it occurs at lower temperatures, and this is what may be expected, as decomposition and consequent reduction in the number of K and Cy ions takes place with weaker solutions at a slower rate.

**Influence of Temperature in Practice.**—It would appear from the foregoing that in practice, by using solutions at high temperatures, there would be a great saving in time or an increase in the extraction. Let us see what really happens. Ores usually contain, besides their gold and silver, one or more metallic minerals, such as iron or copper pyrites, galena, etc., which are also soluble in cyanide solutions, as shown in Table XIX. These become more soluble at higher temperatures, just as gold and silver do. Now, at the lower temperatures a portion of the minerals becomes polarised through the dissolving of the gold and silver and other causes, and that part remains, for the time being, more or less insoluble, while at higher temperatures the effect of polarisation is reduced, and the dissolving area of the minerals is increased. We therefore get a larger quantity of mineral dissolved in the same time, but the gold and silver, being positive, are not polarised, and so their dissolving area is unaltered by rise of temperature. The net result is a greater proportional dissolution of the mineral matter than the gold and silver, and the consequent reduction in the K and Cy ions available for the dissolution of the metals. It is evident, then, that unless the increase in the rate of dissolution of the gold and silver, due to increase of temperature, is more than sufficient to make up for the decrease due to loss of available cyanide, there can be no benefit in using hot solutions.

Again, in practice, when extracting with hot solutions, it is usual to apply

the heated solution to the cold ore. In heating the solution it is deprived of its oxygen, as shown in Table XXII., and on being applied to the ore it is cooled. Now, it has been pointed out that at the higher temperatures the presence of oxygen is of little importance, while at ordinary or low temperatures it is necessary for the dissolution of the gold or silver. Therefore the cooled solution is not in a fit state for the most efficient extractions. From this cause we might expect erratic results.

J. E. Clennell and J. Johnston made, under the direction of Charles Butters,\* a large number of experiments on a working scale with hot and cold cyanide solutions. The hot solutions varied from 100° F. to 140° F., but that selected in most cases was 130° F. when run on, and the cold solution was 70° F. Referring to Table XXII., it will be seen that the rate of dissolution at 70° was about 120 units, and for 130° about 140, or in the ratio of 6:7. This is approximately what we might expect if the solutions were maintained at the same temperature throughout the treatment. As a matter of fact, when leaching on a working scale with hot solutions, it is difficult, if not impracticable, to maintain the same temperature throughout; and further, since the hot solutions decompose in an erratic manner, not altogether dependent on the temperature, it is impossible to maintain the same strengths, in order to draw comparisons. Therefore the net result may be a less effective extraction than if the ore were treated throughout with a cold solution. Clennell and Johnston found that when treating sands and slimes at 130° and 70° the better extraction was at one time in favour of the hot solution, and on a similar sample, at another time was in favour of the cold solution.

**Maximum Dissolution Temperatures.**—From Winkler's results of the solubility of oxygen at different temperatures, it would appear at first sight that when a solution is to be used cold, freezing point should be the most efficient temperature, because we have then a maximum solubility of oxygen. This would be the case were it not that at the lower temperatures the solution becomes more viscous, and thereby offers a greater resistance to the movement of the ions and molecules from one part to another. This increased viscosity hinders the K and Cy ions from getting in contact with their respective electrodes, and it retards the diffusion of the oxygen molecules, and thus lessens the rate of depolarisation. At a point near to freezing, the density of the solution is at a maximum, and by raising the temperature we reduce the viscosity, and therefore the ions and molecules can move with greater velocity. But with rise of temperature the absorption coefficient of oxygen is reduced, and decomposition of the KCy becomes more apparent. A temperature must, however, be reached at which (1) the product of the number of oxygen molecules into their rate of diffusion is a maximum; (2) the product of the number of K and Cy ions into their velocities is a maximum. These two factors are those chiefly concerned in determining the maximum dissolution temperatures for cold solutions.

\* We are indebted to Mr Butters for the particulars.

There must also be a maximum dissolution temperature for hot solutions. This is dependent chiefly on the velocities of the K and Cy ions. The hotter the solution the greater is its fluidity and the less is the resistance to the movement of the ions, but in the case of cyanides a decomposition factor gets introduced which has to be taken into account. The maximum dissolution temperature for hot solutions is then that temperature at which the product of the velocities of the K and Cy ions into their number is a maximum. The amount of oxygen absorbed has always a certain effect, but this is generally not appreciable for the higher temperatures, and vanishes at boiling point.

## CHAPTER XIII.

### DISSOLUTION OF GOLD PHYSICALLY CONSIDERED.

For our purpose in discussing this matter, we may assume that when gold or silver is dissolved in a cyanide solution kept uniform in strength, layers of equal thickness will be dissolved in equal spaces of time.

In an ore, the variety of sizes and shapes of the particles of gold must be very great, but a consideration of two or three assumed cases will assist in understanding what actually takes place in practical operations. (*a*) The extreme case is when the gold is in thin flat plates, so that its surface is constant, or nearly so, until dissolution is complete. It is evident that in this case, under uniform conditions of the solvent, the quantity dissolved will be in direct proportion to the time, and this may be represented by the straight line *A* in fig. 22. Again, when gold is embedded in pyrites so that only one edge or surface can be attacked, it would also have a constant rate of dissolution.

(*b*) The other extreme case is when each gold particle is assumed to be a perfect sphere. Then in each equal interval of time a spherical shell of equal thickness would be dissolved, but each successive shell would be of smaller diameter, and would consequently contain a smaller quantity of gold. Hence in this case the rate of dissolution would be much greater at the beginning than at the end of the operation. Curve *B* illustrates the action for one sphere of any size, or for any number of spheres of equal size.

If, however, in order to approximate more closely to actual conditions, we assume a mixture of spheres of various sizes, it is clear that the smaller spheres will be entirely dissolved before the larger ones, and in such cases the rate at the beginning will be still more increased.

(*c*) The curve *C* has been calculated on the assumption that for each three spheres of diameter 1, there are two of diameter 2, and one of diameter 3.

(*d*) In curve *D* there are one hundred spheres of diameter 1, ten of diameter 2, and one of diameter 3.

The conditions as to relative sizes of particles in actual ores would probably lie between these last two theoretical cases. But as the actual particles of gold are not true spheres, we may expect in practice that the curve would be less convex than *C* or *D*, especially towards the end, because, with any shape except the sphere, the particles would gradually tend towards flatness as they become smaller.

These calculated curves can now be compared with curve *E*, which is

plotted from figures given by H. K. Picard \* of a case at Deloro where one solution was continuously circulated through the ore and cyanogen bromide added at intervals, so that the conditions of solution must have been fairly

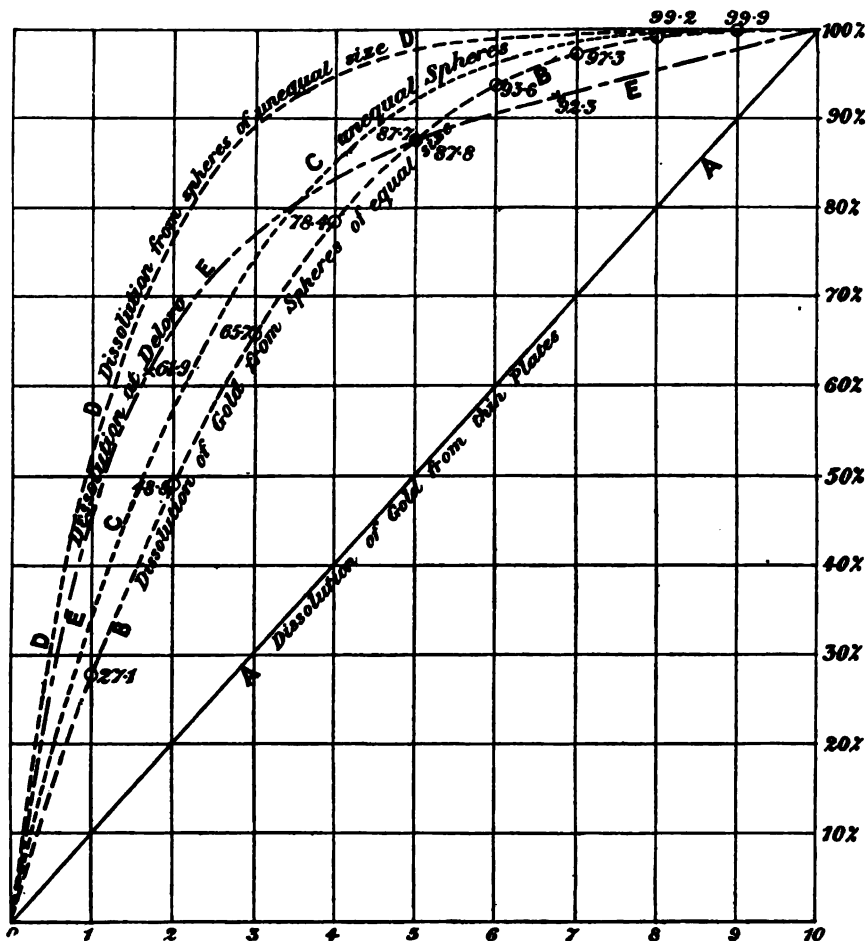


FIG. 22.—Curve *A* shows the rate of dissolution of thin gold plates where the surface exposed to the solution is practically constant. Curve *B* where the gold is in the form of spheres of equal diameter. Curve *C* where the gold is in spheres of unequal size: 1 at 3 units, 2 at 2 units, and 3 at 1 unit diameter. Curve *D* where the gold is in spheres of unequal size: 1 at 3 units, 10 at 2 units, and 100 at 1 unit diameter. Curve *E* is from Picard's figures at Deloro.

constant. The solution was sampled and assayed at intervals. The total gold in the vat was 33.9 ozs., the amount dissolved was 28.6 ozs., and the percentages have been calculated on the latter figure, because we are now considering only the gold capable of being dissolved. It will be seen that

\* *Trans. Fed. Inst. of Min. Eng.*, vol. xv. p. 417, 1897-8.

the curve follows very closely the form to be expected from the above reasoning. Up to an extraction of 80 per cent. it lies between *C* and *D*, and from about 88 per cent. it is almost a straight line.

The figures relating to these curves are given in Table XXIII., in which the time is divided into ten equal parts. The top row of figures opposite each letter shows the percentage of the total gold which has been dissolved during each interval, and the lower figure gives the total percentage dissolved at the end of each period.

This matter has been treated at some length, because it enables a clear mental picture to be formed of the action taking place in a cyanide vat, and leads to one definite reason why in practice the whole of the soluble gold cannot

TABLE XXIII.

	1	2	3	4	5	6	7	8	9	10
A	10% 10	10 20	10 30	10 40	10 50	10 60	10 70	10 80	10 90	10 100
B	27.1 27.1	21.7 48.8	16.9 65.7	12.7 78.4	9.3 87.7	5.9 93.6	3.7 97.3	1.9 99.2	0.7 99.9	0.1 100
C	38.6 33.6	24.0 57.6	16.5 74.1	11.0 85.1	7.0 92.1	4.1 96.2	2.2 98.4	1.1 99.5	0.4 99.9	0.1 100
D	50.2 50.2	26.8 77.0	12.1 89.1	5.7 94.8	3.0 97.8	1.4 99.2	0.5 99.7	0.2 99.9	0.09 99.99	0.01 100
E	45.2 45.2	20.9 66.1	10.7 76.8	6.4 83.2	4.6 87.8	2.9 90.7	2.4 93.1	2.4 95.5	2.3 97.8	2.2 100

*Note.*—Top figure in each square = % dissolved during interval. Lower figure = total % dissolved at end of each interval.

be dissolved. It will be noticed that, except in case *A* (which is purely ideal), more than 90 per cent. of the gold is dissolved in six-tenths of the time required for complete dissolution, and over 95 per cent. in eight-tenths of the total time. So that with poor material it does not pay to give so much extra time for such a small additional recovery.

At Deloro (curve *E*) the gold dissolved was 14 dwts. 18 grs. per ton and the quantity obtained in the last interval was 7.78 grains = 1s. 3½d. per ton, so that it was profitable to carry the treatment to the limit, but with a 5 dwt. material, and with conditions approaching those of curves *B* or *C*, it would be evidently unprofitable to continue the operation beyond seven-tenths of the total dissolution period.

Another set of figures illustrating this point is given by Alfred Chiddy.\* As he does not give the total time required for the complete dissolving of all

\* *Jour. Soc. Chem. Industry*, vol. xix. p. 25.



the accessible gold, the curve cannot be plotted in fig. 22, but is given separately in fig. 23, in which the same general features are apparent, namely,

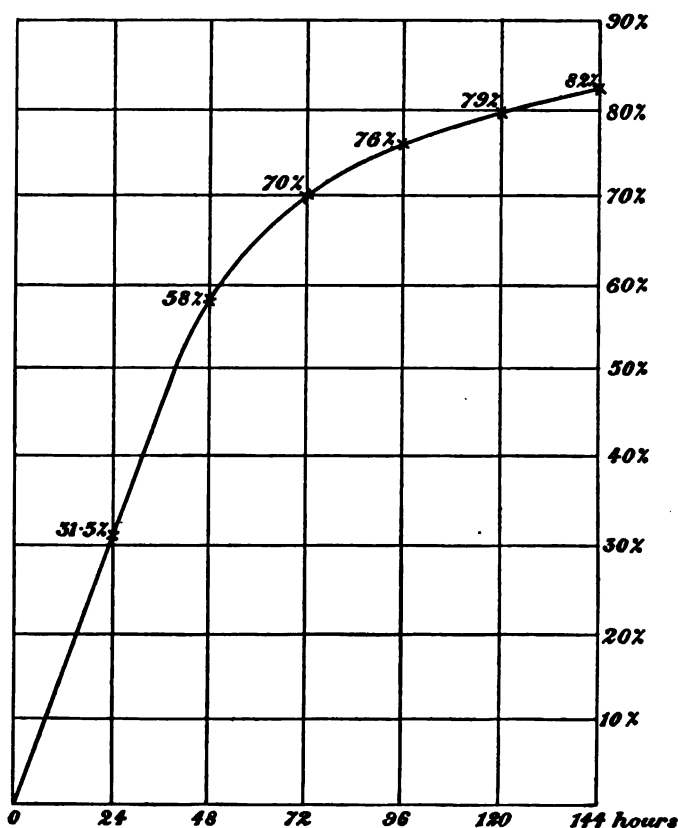


FIG. 23.—Rate of Solubility of Gold in an Ore.

*Note.*—In this diagram the percentages are calculated on the assay value of the ore.

a rapid rise at first followed by a sharp curve, which is succeeded by a slightly inclined straight line.

## CHAPTER XIV.

### ABSORPTION OF AIR BY SOLUTIONS.

**Solubility of Gases in Liquids.**—Gases that do not act chemically on each other or on a solution dissolve in proportion to the pressure of the gas, and when the pressure is lowered or the temperature raised they are expelled. From this it is thought that the process of solution is purely a mechanical one. In general, however, gases dissolved in salt solutions do act chemically to a certain extent. The solvent exerts a selective influence and dissolves more of one gas than another, and when two gases are mixed, each dissolves in proportion to the partial pressure it exerts, combined with its own specific solubility in the solution. Thus air dissolved in water at freezing point and at normal atmospheric pressure contains about 35 per cent. oxygen, whereas air in the atmosphere contains about 21 per cent.

**Solubility of Air in Water.**—When air passes into water, oxygen and nitrogen dissolve until a point is reached at which the number of molecules entering the solution from without in a given time is equal to the number of molecules that fly off. At this point the solution is said to be saturated. The quantity of each gas dissolved is dependent solely on the pressure exerted by the particular gas, and is the same as if only one gas had been present in the space. Thus Bunsen found that the absorption coefficient of oxygen at 0° C. equals 0.04114, and that of nitrogen equals 0.02035. Now, air contains 79 per cent. nitrogen to 21 of oxygen by volume. Hence the partial pressure of the oxygen equals 0.21 of an atmosphere, and that of nitrogen equals 0.79.

Therefore  $0.21 \times 0.04114 = 0.0086394$  proportion of oxygen dissolved, and  $0.79 \times 0.02035 = 0.0160765$  proportion of nitrogen dissolved.

According to these figures, the percentage composition by volume of air dissolved in water at 0° C. is, in round numbers, 65 nitrogen to 35 oxygen.

It is obvious that if we know the absorption coefficients at other temperatures, we can easily find the proportion of each gas dissolved from air.

Table XXIV. gives Bunsen's\* absorption coefficients of oxygen in water from an atmosphere of the gas at different temperatures when the barometer indicates 760 mm. pressure. The figures represent the volume of gas dissolved by one volume of water after the gas had been reduced to 0° C. and 760 mm. pressure. We have calculated from these the oxygen coefficients from air and placed the results in the adjoining column.

The ratio of O coefficient from O to the O coefficient from air is approximately 5 : 1.

\* Bunsen's *Gasometry*.

As a convenient form for practical work we have calculated from Winkler's\* results the weights of oxygen dissolved in water saturated with air, in grains

TABLE XXIV.

Temperature.		Bunsen's O Coeff. from O.	O Coefficients from Air.
C.°	F.°		
0	32	0·04114	0·008639
2	35·6	3907	8205
4	39·2	3717	7805
6	42·8	3544	7442
8	46·4	3389	7116
10	50	3250	6825
12	53·6	3133	6574
14	57·2	3034	6371
16	60·8	2949	6192
18	64	2884	6056
20	68	2838	5959

per ton of 2000 lbs. at 760 mm. pressure and at various temperatures. These are given in Table XXV.

**Solubility of Air in Salt Solutions.**—When a salt is dissolved in water the coefficient of absorption of air is generally smaller, but in the cyanide working solutions we have only small percentages of salts present, which do not materially affect the amount capable of being absorbed. Thus Maclaurin† found that the oxygen coefficient of absorption by a 1 per cent. KCy solution

TABLE XXV.

Temperature.		Grains of O dissolved in 1 ton Water.	Temperature.		Grains of O dissolved in 1 ton Water.
C.°	F.°		C.°	F.°	
0	32	20·37	16	60·8	13·78
2	35·6	19·28	18	64·4	13·22
4	39·2	18·28	20	68	12·7
6	42·8	17·36	22	71·6	12·22
8	46·4	16·52	24	75·2	11·77
10	50	15·74	26	78·8	11·34
12	53·6	15·03	28	82·4	10·92
14	57·2	14·38	30	86	10·51

was 0·0280 at 18° C., whereas that absorbed by water at the same temperature from Bunsen's results is 0·02884. When, however, sea water is employed, or, as in W. Australia, mine water containing 30 per cent. common salt, the absorption coefficient may be materially affected. Thus, from Maclaurin's

\* *Berichte d. d. chem. Gesell.*, xxii. 1773.

† *Jour. Chem. Soc.*, vols. lxvii. and lxviii. p. 211.

figures a 5 per cent. KCy solution reduces the coefficient to 0.0230, and a 30 per cent. KCy to 0.0079. A reduction in the coefficient means a reduction in the amount of available oxygen for purposes of oxidising reducing agents in the solution and ore and of oxidising hydrogen as formed in the process of dissolving the metals.

When the solution is made thick and viscous by means of slime, sand, etc., it has a very marked effect on the coefficient of absorption, although the amount of dissolved salt is very minute. Thus when the pulp consists of two parts of solution (0.05 per cent. KCy) to one part of slime, the coefficient of absorption of air is less than one-hundredth of that of the solution alone.

Another point of practical importance is the time required to saturate the solution with air. If air is removed from a solution by heating or otherwise, the time required to re-charge the solution varies with the viscosity of the solution. An ordinary cyanide solution takes longer than water, and slime pulp takes longer than simple cyanide solution. It often happens that the oxygen of the air absorbed becomes removed by reducing agents, leaving the nitrogen intact, so that, in order to replace the oxygen, air has to be passed into the solution for a considerable time. As a substitute, and to shorten the time of treatment, strong oxidisers are sometimes beneficially added to the pulp.

This increase in the time required to saturate a solution with air points to an increased friction to the gas molecules which retard their diffusion. A rise in temperature lessens the viscosity, and consequently allows a faster rate of diffusion, but rise in temperature diminishes the coefficient of absorption. It is evident, then, that the time of saturation is dependent on these two factors.

## CHAPTER XV.

### ACTION OF VARIOUS CYANIDE SOLUTIONS.

**Relative Dissolving Power of Cyanides.**—The weights of simple cyanides dissolved in equal quantities of water, so as to have the same dissolving effect on gold or silver, are dependent on the ratio of the valency of the base and the molecular weight of the salt. Thus sodium and potassium have a valency of 1. They combine with cyanogen to form NaCy and KCy, which have molecular weights of 49 and 65 respectively. Therefore it will require only 49 parts by weight of NaCy to produce the same dissolving effect as 65 parts of KCy. But in the case of calcium the valency is 2 and the molecular weight of CaCy<sub>2</sub> is 92. Hence it will require  $92 \times 2$  or 184 parts to have the same dissolving effect as 49 of NaCy or 65 of KCy.

In Table XXVI., column 1 gives the formula of various simple cyanides, 2 their molecular weights, 3 valency of the base, 4 relative weights dissolved in equal quantities of water to give the same dissolving effect, 5 relative dissolving power of the same weights of the salts in terms of KCy at 100.

TABLE XXVI.

1	2	3	4	5
NH <sub>4</sub> Cy*	44	1	44	147·7
NaCy	49	1	49	182·6
KCy	65	1	65	100
MgCy <sub>2</sub>	76	2	152	42·7
CaCy <sub>2</sub>	92	2	184	35·3
SrCy <sub>2</sub>	139·5	2	279	23·3
BaCy <sub>2</sub>	189	2	378	17·2

The stability of these compounds varies greatly, KCy being the most stable, and CaCy<sub>2</sub> the least. The following is the order of stability in an atmosphere containing CO<sub>2</sub>—

KCy, NaCy, NH<sub>4</sub>Cy, MgCy<sub>2</sub>, BaCy<sub>2</sub>, SrCy<sub>2</sub>, CaCy<sub>2</sub>

**Dissolving Power of Potassium Zinc Cyanide.**—The comparative dissolving power of potassium zinc cyanide and potassium cyanide has been much discussed. It was thought by many in the early days of the cyanide process that the K<sub>2</sub>ZnCy<sub>4</sub> formed in the precipitating of the gold by zinc, was useless for dissolving any further quantity of gold from the ore. It was, however,

\* NH<sub>4</sub>Cy has an abnormally large dissolving effect on silver, that given being for gold.

soon pointed out that  $K_2ZnCy_4$  had a solvent action on gold in ores. W. R. Feldtmann\* says, "Addition of alkali to working cyanide solutions which have become somewhat weak in alkali, brings up the strength by regenerating, i.e. decomposing, the zinc cyanide . . . so that, as a matter of fact, when the solutions are pretty strongly alkaline they contain no zinc as cyanide, but only the hydrate dissolved in alkali." He further says that the African Gold Recovery Company made some tests which proved that potassium zinc cyanide has a solvent action on gold, even without the presence of free KCy.

G. A. Goyder† prepared some pure  $K_2ZnCy_4$ , and found that gold dissolved in it in presence of oxygen with the production of gold potassium cyanide and oxide of zinc. A number of other authorities have also shown that  $K_2ZnCy_4$  dissolves gold.

We made experiments to compare the dissolving effects of KCy and  $K_2ZnCy_4$  on gold. Some '98' per cent. KCy was dissolved in water, and its dissolving effect on gold was determined. ZnO was added to a portion of the same solution in excess until it was probably saturated with zinc, and the dissolving effect of this solution on gold was also determined. These were found to be in the ratio of 109:96 respectively. A large excess of alkali was next added to both solutions, and their dissolving effects were found with KCy to have increased to 133, and with  $K_2ZnCy_4$  to 128. It is clear from these results that the presence of zinc in the solution is not very detrimental, and since the alkali added to the KCy, increased its dissolving effect in nearly the same proportion as that of  $K_2ZnCy_4$ , it would appear that this increase in the dissolving effect was due to other causes than the formation of zincate of potash and free KCy, as above stated.

#### MIXED SOLUTIONS.

It is usual in the cyanide process to regard the solution as consisting of potassium cyanide, whereas in reality it is a mixture of a number of substances which form a solution of a very complex nature. Some of these substances play no part in the dissolution of the gold and silver, while others accelerate or retard the process, either directly or indirectly.

The presence of practically inert substances as  $K_2SO_4$ ,  $KCyS$ , etc., can cause little or no increase in the solubility of the gold and silver, but has rather the reverse action, through increasing the viscosity of the solution, and thereby decreasing the velocities of the K and Cy ions of the KCy salt.

In the working of the process it is found that salts do not accumulate sufficiently to prove very injurious. The following analysis by Philip Argall‡ gives some idea of the changes that take place in ordinary KCy solutions when zinc precipitation is employed. The solutions were originally made from the

\* *Eng. and Min. Jour.*, lviii. pp. 218-219, 1894.

† *Chem. News*, vol. 72, p. 96.

‡ *Eng. and Min. Jour.*, part ii. p. 249, 1897.

ordinary KCy, and the strength was kept up by the addition of this salt from time to time. The results are given in Table XXVII., as per cent.

A is the analysis of the solution before precipitation, B is the same solution after passing through the zinc boxes. This solution had been used continuously for six months, during which time it had passed through 11,000 tons of ore. C is the same solution after thirteen months' use, during which time 26,000 tons of ore had been treated. D is the same solution after precipitation.

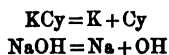
TABLE XXVII.

	A	B	C	D
KCy, . . . . .	0.501	0.491	0.505	0.505
HCy, . . . . .	0.061	0.079	0.017	0.021
Total Single Cyanides, . . . . .	1.302	1.335	1.470	1.445
K <sub>2</sub> FeCy <sub>6</sub> , . . . . .	0.098	0.117	0.018	0.026
KCyS, . . . . .	0.212	0.207	0.058	0.057
Zn, . . . . .	0.316	0.360	0.363	0.388
CaO, . . . . .	0.085	0.082	0.172	0.172
Gold in ozs., . . . . .	1.15	0.087	1.82	0.08

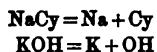
In some cases reducing agents have been known to accumulate to such a degree as to prove very detrimental, by retarding the dissolution of both gold and silver. When the KCy solution is very weak, the presence of small quantities of soluble sulphides have a direct effect on silver, by forming a superficial coating of silver sulphide on the particles, which renders the silver almost insoluble. With strong solutions this effect is not marked.

On the other hand, oxidising agents in suitable quantities have a beneficial effect and tend to keep the rate of dissolution of the metals more constant. In fact, without an oxidiser in some form, dissolution soon ceases under ordinary atmospheric changes. Then, again, the presence of substances in which AgCy and AuCy are soluble, as thiosulphates of the alkalis, ammonia, etc., tend to increase the solubility of gold and silver in cyanide solutions, but these substances sometimes interact with the cyanide, so that their value may be considered as doubtful.

The solubility of gold in alkaline cyanides varies with the base, as already stated, and it follows that the amount of gold that will dissolve in a given time in a 1 per cent. KCy solution will be different in a 1 per cent. NaCy solution, but if we deal with normal solutions or equally diluted normal solutions their effects will be the same. When equal quantities of normal solutions of KCy and NaOH are mixed and moderately diluted, they will have exactly the same action as normal solutions of NaCy and KOH diluted to the same extent. In dilute solutions these compounds become completely dissociated into ions thus—



and in the second case,



It will be seen at a glance, in both cases, that we have the same K and Na cations and the same Cy and OH anions. It has been proved that the action of dissociated solutions depends only on their ions, and since the ions in both cases are alike, the action in both cases must be the same.

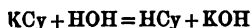
When two solutions are mixed which do not alter the properties of each other, such as KCy and NaCy, a metal dissolves in them, and divides itself between the two solutions in the ratio of their respective ions. Thus, in dissolving gold in a mixture of KCy and NaCy, we get formed  $\text{KAuCy}_2$  and  $\text{NaAuCy}_2$  in proportion to the ions present of the respective salts.



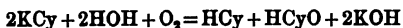
## CHAPTER XVI.

### SOURCES OF LOSS OF CYANIDE.

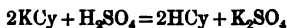
**Decomposition of the Solution.**—An alkaline cyanide is very easily decomposed by any mineral acid, and less easily by salts of these acids. Even water decomposes single cyanides by hydrolytic action. Potassium cyanide is an inodorous substance in an atmosphere free from moisture, but when moisture is present the well known odour of HCy is at once recognised. The moisture acting as an acid produces the following chemical changes—



and in presence of dissolved oxygen



Hydrocyanic acid being only slightly soluble in water, and generally less soluble in solutions of salts, soon begins to escape into the atmosphere. True, the decomposition of KCy in this way is infinitesimally small as compared with the action of other acids. Even a weak acid, such as carbonic acid, has a stronger action by thousands of times, while the highly dissociable acids, as HCl,  $\text{NH}_4\text{OH}$ , and  $\text{H}_2\text{SO}_4$ , have such powerful actions that in a moderately concentrated state they will split up even such a slightly dissociable substance as cyanogen, forming compounds not belonging to the cyanogen group. If, however, the acid is dilute, such as we find present in partly decomposed ores, the action is similar to that which occurs with water, thus—



In very dilute solutions the HCy escapes into the air so very slowly that, even some hours after the solution became acid, practically the whole of the HCy is still present, and if alkali is added, it will be found that the solution has lost little of its original strength. Some HCyO or KCyO is also formed in presence of dissolved air.

It often happens on heating KCy solutions much above  $140^\circ \text{F.}$ , and even at lower temperatures, formates and acetates are formed, and a number of complicated changes occur which involves loss of cyanide. The reactions are, however, very uncertain, and it sometimes happens that the solution may be raised even to boiling point, without showing any appreciable chemical change.

In the absence of reducing agents, oxygen absorbed from the air slowly oxidises the cyanide to cyanate or cyanic acid, as shown above. In working solutions, traces of reducing agents are nearly always present, so that the loss

from this cause is very small. Oxidising agents, such as  $\text{KMnO}_4$ ,  $\text{K}_2\text{FeCy}_6$ ,  $\text{PbO}_2$ , etc., readily oxidise cyanide to cyanate, but in working solutions, and especially in presence of reducing agents, this action is not marked.

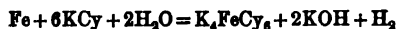
Vasculose readily destroys cyanide in absence of an excess of alkali. Coconut fibre, jute, and wood are especially destructive when new, but after having been in contact with the solution some time they lose this effect. Vasculose acts as a reducing agent in a cyanide solution.

Carbonic acid is absorbed by cyanide solutions from the atmosphere, and in absence of excess of free alkali  $\text{HCy}$  is evolved. In moderately still air a square yard of surface absorbs 1000 c.c. of  $\text{CO}_2$  per hour, and in a strong wind or by constantly stirring it absorbs 1200 to 1300 c.c. per hour. The rate of absorption is hardly affected by the amount of alkali present within working limits, but the tension of the  $\text{CO}_2$  has a very marked influence. The figures given refer to cases where the top of the tanks is several feet above ground level. But when the top of the tanks is on the level or below the level of the surrounding ground, the rate of absorption may be three or four times as fast.

The presence of soluble and insoluble carbonates facilitates decomposition of cyanides.

**The Action of Cyanide on the Constituents of the Ore.**—The cyanide process is only made possible by the fact that the action of the solution on the ore constituents, compared with that on gold and silver, is for the most part but slight. That is to say, that the gold and silver are electro-positive in a cyanide solution to the other matter with which they are associated. A large number of the constituents of ores are, however, appreciably dissolved in cyanide solutions, and since they expose generally very large surfaces compared to that of the gold and silver, it often happens that much larger quantities are dissolved. The following are a few of the ore constituents most frequently met with, and their effects on the solution.

**Metallic Iron or Steel** usually gets introduced into the ore in the process of crushing and becomes disseminated throughout the mass in fine grains, and is employed in the construction of parts of the plant. Iron in a  $\text{KCy}$  solution dissolves slowly and is much less soluble than gold or silver, as shown in fig. 18. The final reaction is probably expressed by the equation—



The hydrogen becomes oxidised to water, as formed, by oxygen or oxidising agents in the solution.

The actual weight of iron introduced into the ore or brought into contact with the solution is far in excess of the gold and silver usually present, but the gold and silver being in a finer state of division and being more soluble, it is probable that a larger weight of these metals dissolve during the time of treatment. But it must be remembered that 56 parts of iron consume 390 parts of  $\text{KCy}$ , whereas 197 of gold consume only 130 parts of  $\text{KCy}$ . That is to say, 1 lb. of iron requires 7 lbs. of  $\text{KCy}$  for dissolution, while 1 lb. of gold requires only 0.66 lb.  $\text{KCy}$ , or less than one-tenth.

Iron not only consumes much cyanide, but it also consumes oxygen required for the dissolution of the gold and silver, and it follows from the reactions that 56 parts of iron will consume twice as much oxygen as 197 parts of gold. (See page 66.)

Ferric oxide in electrical contact with gold in a cyanide solution acts as a depolariser, and produces a lower oxide that is more soluble in the solution.

**Iron Sulphides** are very generally present in gold ores in two well known forms, as marcasite (the orthorhombic mineral) and pyrite (the isometric mineral); both have the same composition, and are represented by the formula  $\text{FeS}_2$ . *Pyrites* is the general name for them.

*Marcasite* is of a paler and duller colour and lower specific gravity (4.8) than pyrite; also it is well known to have a greater tendency to oxidise in moist air, with formation of  $\text{FeSO}_4$ . *Pyrite* is of a brassy yellow colour, resists atmospheric oxidation much more than marcasite, and has a specific gravity of about 5. It is very common to find both minerals associated together in gold ores.

In a cyanide solution marcasite is much less soluble than pyrite, as may be seen on reference to fig. 18. The reaction appears to be doubtful, but the chief compounds formed are  $\text{K}_4\text{FeCy}_6$ ,  $\text{KCyS}$ ,  $\text{K}_2\text{S}$ , and  $\text{K}_2\text{S}_2\text{O}_8$ . From several experiments we have made, it is evident a very considerable quantity of oxygen is consumed in the reaction, as we find that pyrites is hundreds of times more soluble when a large excess of oxygen is present. Unoxidised pyrites appears to be very insoluble in absence of oxygen, at atmospheric pressure and ordinary temperatures. With an excess of oxygen the pyrites caused the solution to become coloured, varying in shade from an amber tint to a muddy brown. It loses colour by standing when exposed to the atmosphere and sunlight for some days, or by the use of oxidising and desulphurising agents. This brown solution was found to have lost much of its dissolving power on gold and silver, but the power is easily restored by the use of desulphurising and oxidising agents.

It has long been observed that when marcasite has been allowed to weather, it heats and rapidly oxidises, forming largely  $\text{FeSO}_4$  and  $\text{H}_2\text{SO}_4$ . On the other hand, pyrite oxidises slowly to limonite,  $\text{Fe}_2\text{O}_3(\text{OH})_3$ , and sulphur for the most part, with but little  $\text{FeSO}_4$ . Caldercott\* says that Dr Lœvy proved that when  $\text{FeS}_2$  is gradually oxidised the first compound formed is  $\text{SO}_2$ , also that  $\text{H}_2\text{S}$  is formed through the action of  $\text{H}_2\text{SO}_4$  on  $\text{FeS}$ , the latter being formed by the oxidation of one atom of sulphur of the  $\text{FeS}_2$ .

The presence of  $\text{FeSO}_4$  in an ore is the cause of a very considerable consumption of cyanide. It is soluble in water, and if the ore is leached with clean water a portion may be removed, which thus saves cyanide. Some, however, always remains behind, and this becomes neutralised on the addition of alkali, forming ferrous hydrate and alkaline sulphate. Ferrous sulphate and hydrate act as deoxidisers and consume oxygen that might otherwise be of service in dissolving the gold and silver. Ferrous sulphate oxidises to

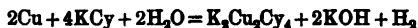
\* *S. African Min. Jour.*, Oct. 23, 1897, from paper read before S. Af. Chem. and Met. Soc.

normal and basic ferric sulphates. The former is soluble in water, and the latter is soluble in a solution of the former. They are both acted on readily by KCy solutions, forming, amongst other compounds,  $K_4FeCy_6$ , some Prussian blue, and probably  $K_3FeCy_6$ . Metallic iron in the ore reduces some ferric sulphate to ferrous salts.

Ferrous hydrate is readily oxidised to ferric hydrate, a name applied to many indefinite compounds, having the formula  $Fe_2O_3 \cdot xH_2O$ , of which the exact composition varies. Some of these compounds are very insoluble in KCy, but others dissolve more or less readily, probably forming  $K_4FeCy_6$ , and acting to some extent as oxidisers.

*Limonite*, a mineral often associated with gold ores, is a hydrated oxide of iron. It does not appear to dissolve in KCy to any appreciable extent, but often forms unstable suspensions that become troublesome in the process of leaching.

Copper in the metallic state is found occasionally associated with gold ores, and usually it is introduced during the mining operations in the form of detonator cases. The final reaction when copper is dissolved in KCy is generally stated as



Therefore 1 part of copper requires 2 parts KCy for solution, or three times as much as gold. The solution also loses oxygen by its combining with some of the hydrogen formed at the negative electrode.

Copper sulphides are invariably present in gold-bearing ores, the amounts varying from a trace up to the proportions of a rich copper ore. *Copper pyrites* is perhaps the most commonly met with. *Erubescite*, *covellite*, and *redruthite* are less often present.

These minerals are all acted on by cyanide solutions, but some are much more soluble than others. Tests to determine the solubility of a mineral by itself are only of partial value, as much depends on the other minerals with which it is in contact. Thus copper pyrites, which by itself dissolves slowly in KCy, when in contact with a less negative mineral, as *marcasite*, dissolves much more rapidly. Then again, when iron pyrites is present and becomes oxidised to ferric sulphate, most copper sulphides are also oxidised to sulphate. The copper sulphate remains in solution, but when this compound comes into contact with metallic iron the copper precipitates, and this precipitate readily dissolves in cyanide solutions.

The presence of copper is not necessarily a bar to the treatment of an ore by cyanide, for it sometimes happens that when even 2 or 3 per cent. of copper is present an ore can be treated profitably by cyanide, but not by other processes. On the other hand, sometimes an ore containing less than 0.5 per cent. copper could not be treated profitably, owing to the destruction of cyanide. A mere chemical analysis of an ore is not to be relied on as an absolute guide in testing copper-bearing ores, as much depends on the physical conditions in which the copper and gold exist.

There are a very large number of minerals containing copper in some form associated with ores that carry both gold and silver, and it sometimes happens that the gold and copper are dissolved, but scarcely any of the silver, while in others the gold is practically the only metal dissolved. It is probable that the copper and silver are sometimes in chemical combination, as in the minerals *stromeyerite*,  $\text{Cu}_2\text{SAg}_2\text{S}$ , and *jalpaite*,  $\text{Cu}_2\text{S}_2\text{Ag}_2\text{S}$ , and at others are mixtures of two or more simple compounds, as  $\text{CuS}$  and  $\text{AgS}$ . The solubility of silver is usually small when copper is present, whereas the solubility of gold is not generally affected to anything like the same extent.

Occasionally the minerals *malachite*,  $\text{CuCO}_3\text{CuH}_2\text{O}_2$ , and *azurite*,  $2\text{CuCO}_3\text{CuH}_2\text{O}_2$ , are found associated with gold and silver ores. These consume so much cyanide, that ores containing even very small quantities of either become unprofitable for cyanide treatment.

**Native Arsenic** is occasionally found associated with gold ores, or is reduced from its salts in the process of crushing. Arsenic does not interact with potassium cyanide nor with alkali in the solution, but nearly all compounds of arsenic found in gold ores are soluble in caustic alkali and decompose  $\text{KCy}$  in absence of free alkali.

**Mispickel** (arsenical pyrites),  $\text{FeAsFeS}_2$ , is, next to iron pyrites, perhaps the most abundant metallic compound found associated with gold ores, and is in general the more auriferous of the two. Mispickel is acted on by cyanide, its relative solubility to gold being very small, as shown in fig. 18. It weathers in moist atmospheres, for the most part to iron sulphate and to hydrate and oxide of arsenic. The latter does not combine with the cyanide, but in absence of free alkali decomposes the solution, liberating  $\text{HCy}$  and combining with the alkali.

In treating mispickel ores, it is generally found that the addition of large amounts of lime to the ore effects a considerable saving in cyanide and improves the extraction. If magnesia is procurable it will be found to answer equally well, and as a rule a higher extraction can be relied on.

Sulphides of arsenic, as realgar and orpiment, are both attacked by alkalies, forming arsenites and thio-arsenites. Alloys of iron and arsenic, as arsenical iron, are occasionally rich in gold and silver. These compounds are little acted on by cyanides, but the gold and silver are not readily attacked unless the mineral is reduced to a very fine state of division.

The compounds of alkali and arsenic that form act as reducers, and consume oxygen dissolved in the cyanide solution, and in this way retard the dissolution of the gold and silver.

**Antimony Sulphide**, as stibnite, is often found associated with gold ores, and is sometimes highly auriferous. Like arsenic, antimony does not form any definite compound with cyanide, but the sulphide is very soluble in caustic alkali and decomposes cyanide, combining with the alkali, and forming antimonite and thio-antimonite, also some  $\text{KCyS}$  is formed and  $\text{HCy}$  is evolved. The antimony compounds act as strong de-oxidisers, and remove the absorbed oxygen from the solution by forming antimonate and thio-

antimonate. Gold and silver may be rendered almost insoluble from this cause, and also from the fact that much of the cyanide is destroyed.

Other antimony minerals, as antimony bloom, senarmontite, and antimony ochre, occur in gold ores, but much less frequently than stibnite, and are less difficult to deal with.

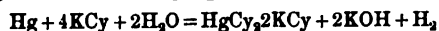
**Tellurium** is often found in minerals associated with gold ores, which are usually highly auriferous and argentiferous. In some minerals it would appear to be in the form of an alloy with the gold or silver, or in more or less definite compounds, as calaverite, krennerite, petzite, and sylvanite, forming tellurides of gold and silver. Nagyagite, or foliated tellurium, is a rather complex substance, containing, besides gold and silver, sulphur, lead, copper, antimony, etc., in varying quantities. Gold is sometimes found in such minerals as hessite and melonite, and tellurium is also found in auriferous arsenical iron, mispickel, iron and copper pyrites.

The presence of tellurides, when forming any appreciable proportion of the metallic minerals, usually makes an ore difficult to treat, as the action of the cyanide on the gold is slow. The cause of this slowness of action is not altogether apparent, but it is very evident that there is a wide difference in the solubility of the gold in different telluride minerals, and even in the same minerals from different localities. Thus in Western Australia, where the gold is in a sulpho-telluride, the ore is capable of being dealt with by fine grinding and long treatment, whereas the same treatment applied to certain Mexican and United States tellurides extracts little or no gold.

There is no action between the cyanogen radical and the tellurium, but the alkali in presence of oxygen invariably dissolves some of the metal, and also acts on the sulphur when in combination, forming a solution which has a reducing action. When a telluride ore is roasted it leaves a residue containing  $\text{TeO}_2$ , and this oxide is very soluble in  $\text{KOH}$ , forming a tellurite, which also acts as a reducing agent and absorbs oxygen from the solution. The same change takes place with  $\text{KC}_y$ , with evolution of  $\text{HC}_y$ . Roasted tellurides are, however, capable of being treated and the gold extracted with good results.

**Mercury** as cinnabar,  $\text{HgS}$ , is practically the only mineral of mercury to be met with in gold ores. Its action on cyanide solution is *nil*. Recently, however, mercury has been found in Kalgurli ores as coloradoite, which is readily attacked by cyanide. Metallic mercury is always present in tailings, being introduced from the amalgamation process, and is sometimes in considerable quantity. In a well managed mill the loss in mercury usually varies from 0.01 to 0.03 lb. per ton of ore crushed, but this is often exceeded when much base mineral is present. Theoretically, the whole of this finds its way into the cyaniding vats, and is acted on by the cyanide solution.

Mercury is somewhat less soluble than gold in cyanide, and probably combines according to the following equation—



Oxygen is required for the dissolution, in order to combine with the hydrogen as formed, and this is taken from the oxygen absorbed by the solution. Thus 200 parts of mercury require 260 parts of KCy for solution, that is, twice as much as gold requires, and they also require twice as much oxygen.

Mercury has, however, a far greater affinity for sulphur than for cyanogen, and as the solutions usually contain sulphides, thiosulphates, and thiocyanates, these or the sulphur in these combine with mercury to form  $\text{HgS}$  and a number of complex compounds, amongst which are  $\text{Hg}(\text{SCy})_2\text{KCyS}$ ,  $\text{HgCy}_2\text{KCyS}$ , etc., which do not appear to retard the dissolution of the gold and silver. Mercury, therefore, acts as a desulphurising agent, and although it consumes cyanide, its presence is not altogether undesirable.

**Zinc**, as zinc blende, is not commonly met with in gold ores. When taken fresh from the mine in an unoxidised condition it is only slightly acted on by KCy, forming some KCyS and probably some complex substances. Blende from some mines is much more soluble than from others, and this appears to be somewhat dependent on the minerals associated with it. If partly oxidised, such as happens when ore weathers, blende destroys both cyanide and alkali.

**Galena** is often associated with both gold and silver ores. In a clean unoxidised condition it is only slightly acted on by KCy, but with long contact KCyS is formed. When partly oxidised by atmospheric influences, long contact with KCy produces evolution of HCy.

Minerals taken from different localities vary very much in the way they are acted on by KCy solutions, as a great deal is dependent on the other minerals with which they are associated. Thus, marcasite or pyrite acted on separately have comparatively small effect on a KCy solution, but when the two minerals are mixed and dealt with together the destruction of cyanide is much increased.

Again, many of the complex compounds formed with the KCy are decomposed in the precipitating boxes, the metal being deposited or forming insoluble compounds which precipitate, while others are rendered less soluble, and probably become to some extent precipitated in the ore.



## CHAPTER XVII.

### PRECIPITATION OF THE GOLD AND SILVER.

#### *Section I.*

**Electrolytic Methods.**—It is usual in the cyanide process to refer to precipitation by an electric current, applied from an external source, as being distinct from that of precipitation by the so-called galvanic couple. But on investigation it will be found that the changes, whereby precipitation is effected, are very similar. In the precipitation of the metal, it is necessary that the current should reach a certain minimum electro-motive force, which varies with different metals, with the same metal in different solutions, and to some extent with the strength of the solution. The current with a suitable E.M.F. may be supplied from an external source, such as from a dynamo or battery, when the E.M.F. and current strength may be measured and regulated with great accuracy. Thus, if we pass a current between electrodes in a gold cyanide solution with an E.M.F. of 1 volt, and find that this is not sufficient to precipitate the gold, we can readily increase it to 2 or 3 volts, or until the precipitation appears satisfactory.

**Precipitation by Galvanic Couples.**—The current may also be supplied by immersing in the solution a metal which has a difference of potential at two parts of its surface, *i.e.* a galvanic couple. But we have not then the same facilities for regulating the E.M.F. as when the current is supplied from an external source. Thus, silver containing lead as an impurity has a difference of potential between the positive silver and the negative lead, which in a cyanide solution generates a current, but this current is not of sufficiently high E.M.F. to precipitate gold from this solution. If, now, we immerse in the solution a piece of commercial zinc which contains lead, a current is generated between the positive zinc and the negative lead of a sufficiently high E.M.F. to decompose the gold solution and precipitate the metal. In this way we are able to determine which combinations generate a current of the requisite E.M.F. The minimum E.M.F. to precipitate the gold varies with the strength of the solution, with the temperature, and, as we shall see later, with the physical condition of the surface of the negative metal.

When the gold is precipitated it asserts its own potential, and tends to re-dissolve, producing an E.M.F. which tends to drive a current in the opposite direction to that of the precipitating current. The result is that the E.M.F. of the latter is reduced by that of the gold. Thus Christy\* found zinc to have

\* See Table XXIX.



a potential of  $+0.82$  volt, and lead to have a potential difference of  $+0.05$  volt, with respect to a  $0.65$  per cent. KCy solution. Therefore the E.M.F. of a zinc-lead couple in that solution equals  $0.82$  minus  $0.05$ ,\* that is,  $+0.77$  volt. The lead in time becomes covered with gold, and as the potential difference of gold, with respect to the same solution, was found to be  $+0.23$ , the E.M.F. of the current falls to  $0.82$  minus  $0.23$ , or  $+0.59$  volt. We do not get the full benefit of the  $0.77$  voltage, for at the moment gold deposits on the lead, we form a gold-lead couple, whereby the gold tends to re-dissolve with a varying E.M.F., which ultimately becomes  $0.23$  minus  $0.05$ , or  $0.18$  volt. This is small comparatively, and for that reason lead is a suitable metal on which to deposit gold. On the other hand, if the negative metal was platinum, we should have an ultimate E.M.F. of  $-0.46 + 0.23$ , or  $0.69$  volt, which would cause the precipitated gold to re-dissolve about four times as fast as in the case of lead. If the negative electrode has a higher potential than gold, as for instance tin, then when gold is precipitated, we have two couples acting in the same direction, *i.e.* zinc-gold and tin-gold, but as the difference of potential between tin and gold is  $0.24$  minus  $0.23$ , or  $0.01$  volt, it follows that a zinc-tin couple is suitable for precipitating gold from this solution. It is therefore important, when precipitating metals from solutions by galvanic couples, that the electro-negative metal should stand in the electro-chemical series as near to the metal being precipitated as practicable.

The above results refer to a  $0.65$  per cent. of pure potassium cyanide solution, but when a weaker or stronger solution is used, or, as in practice, a mixture of several solutions, the results vary accordingly.

**The Electro-chemical Series.**—There is a definite difference of potential between a metal, or metallic mineral, with respect to the solution in which it may be immersed. In cyanide solutions, this potential difference is not always the same, but varies with the strength, and may sometimes be positive and sometimes negative, according to the substances, as already shown by example. When a positive metal dissolves, the transition into ions is attended with a gain of energy. The converse occurs when the ions are transformed into metal (precipitated).

Professors A. Von Oettingen† and S. B. Christy‡ have made a large number of determinations of potential differences of substances with respect to cyanide solutions of different strengths. We reproduce these in Tables XXVIII. and XXIX. Von Oettingen's final observations are given, but we have roughly spaced them according to the numerical values, so as to show the changes of potential more forcibly. Christy's results are reproduced and tabulated as in his original paper.

These tables are useful for making comparisons, but as the results were

\* The total E.M.F. is the algebraic difference of the two potentials. Thus, if two metals have the same signs, the smaller potential must be deducted from the greater, but if of opposite signs, they must be added together.

† *Jour. Chem. and Met. Soc. S. Africa*, Feb. 1899, pp. 21–28.

‡ *Trans. Am. Inst. Min. Eng.*, Sep. 1899.

obtained with pure cyanide solutions, they must not be taken as strictly true for working solutions found in actual practice. Potentials obtained as the above are necessarily, owing to experimental difficulties, the average results of a number of observations for each substance, and this accounts to some extent for differences in the figures of the two authorities.

TABLE XXVIII.—*Von Oettingen's Electro-motive Series of certain Metals in KCy Solutions.*

Last Observations.							
N. 1000 KCy = '0065 per cent.		N. 100 KCy = '065 per cent.		N. 10 KCy = '65 per cent.		N. 1 KCy = 6·5 per cent.	
Ni	-0·560	Fe <sub>2</sub> O <sub>3</sub>	-0·750	Fe <sub>2</sub> O <sub>3</sub>	-0·720	Fe <sub>2</sub> O <sub>3</sub>	-0·700
Au	-0·474	Ni	-0·488	Ni	-0·392		
Cu	-0·230					Fe	-0·146
Ag	-0·200	Au	-0·056				
		Hg	-0·056	Fe	-0·012		
		Ag	-0·020				
PbO <sub>2</sub>	-0·006	Fe	+0·022	Hg	+0·024		
Fe	+0·050	PbO <sub>2</sub>	+0·070	Co	+0·118		
				PbO <sub>2</sub>	+0·118		
Pb	+0·120	Pb	+0·120	Pb	+0·128	PbO <sub>2</sub>	+0·160
				Ag	+0·176	Pb	+0·164
						Ni	+0·194
		Co	+0·240	Au	+0·218	Co	+0·196
		Cu	+0·380			Hg	+0·200
Zn	+0·480	Zn	+0·604	Cu	+0·648	Au	+0·306
				Zn	+0·800	Ag	+0·314
						Cu	+0·924
						Zn	+0·940

Taking Von Oettingen's figures for a solution of 0·65 per cent. KCy, the following are some of the combinations, with their numerical values:—

$$\begin{array}{llll}
 a \left\{ \begin{array}{l} \text{Zn} \\ \text{Cu} \end{array} \right. \begin{array}{l} +0\cdot800 \\ +0\cdot648 \end{array} & b \left\{ \begin{array}{l} \text{Zn} \\ \text{Ag} \end{array} \right. \begin{array}{l} +0\cdot800 \\ +0\cdot176 \end{array} & c \left\{ \begin{array}{l} \text{Zn} \\ \text{Pb} \end{array} \right. \begin{array}{l} +0\cdot800 \\ +0\cdot128 \end{array} & d \left\{ \begin{array}{l} \text{Zn} \\ \text{Au} \end{array} \right. \begin{array}{l} +0\cdot800 \\ +0\cdot218 \end{array} \\
 \hline
 +0\cdot152 & +0\cdot624 & +0\cdot672 & +0\cdot582
 \end{array}$$

Now, as the value for gold in the same column is +0·218, and assuming that this will measure the actual reverse E.M.F., due to the gold when it deposits on the cathode or negative metal, we should have to form a couple

with a greater E.M.F. than 0.218 in order to precipitate gold. The result  $\alpha$  shows that a zinc-copper couple develops an E.M.F. of only 0.152, and if the above assumption were true, no gold should be precipitated. As a

TABLE XXIX.—*Christy's Electro-motive Series of Metals and Minerals in KCy Solutions.*

	$\frac{M}{1}$ KCy =6.5 p. ct. volts.	$\frac{M}{10}$ KCy =0.65 p. ct. volts.	$\frac{M}{100}$ KCy =0.065 p. ct. volts.	$\frac{M}{1000}$ KCy =0.0065 p. ct. volts.
Aluminium, . . . . .	+0.99	+0.90	+0.76	+0.40
Zinc, amalgamated, . . .	+0.93	+0.82	+0.70	+0.44
Zinc, commercial, . . .	Not determ.	+0.77	+0.59	+0.39
Copper, . . . . .	+0.81	+0.62	+0.37	+0.16
Cadmium, . . . . .	+0.61	+0.57	+0.35	...
Cadmium, amalgamated, .	+0.55	+0.31	+0.19	...
Tin, . . . . .	+0.45	+0.24	+0.17	+0.06
Bornite, . . . . .	+0.45	+0.25	-0.18	...
Copper, amalgamated, . .	+0.39 (?)	+0.41	-0.14 (?)	-0.12 (?)
Gold, . . . . .	+0.37	+0.23	+0.09	-0.38
Silver, . . . . .	+0.33	+0.15	-0.05	-0.36
Copper-Glance, . . . . .	+0.29 (?)	+0.25	+0.05	-0.44
Lead, . . . . .	+0.13	+0.05	+0.01	...
Tin, amalgamated, . . .	Not determ.	+0.01	-0.07	-0.12
Lead, amalgamated, . . .	Not determ.	...	-0.03	...
Quicksilver, . . . . .	-0.09	+0.01	-0.11	...
Gold, amalgamated, . . .	...	...	-0.18	-0.26
Antimony, . . . . .	+0.06	+0.03	-0.03	...
Arsenic, . . . . .	+0.04	-0.05	-0.21	...
Bismuth, . . . . .	+0.00	-0.06	-0.20	...
Niccolite, . . . . .	-0.11	-0.17	-0.44	...
Iron, . . . . .	-0.17	-0.24	-0.24	...
Chalcopyrite, . . . . .	-0.20	-0.34	-0.44	...
Pyrite, . . . . .	-0.28	-0.42	-0.48	...
Galena, . . . . .	-0.28	-0.48	-0.52	...
Argentite, . . . . .	-0.28	-0.56	-0.55 (?)	...
Berthierite, . . . . .	-0.30	-0.52	-0.52	...
Speisscobalt, . . . . .	-0.30	-0.33	-0.50	...
Magnetopyrite, . . . . .	-0.30	-0.40	-0.54	...
Fahlore, . . . . .	-0.36	-0.52	-0.52	...
Arsenopyrite, . . . . .	-0.40	-0.45	-0.54	...
Platinum, . . . . .	-0.40	-0.46	-0.50	...
Cuprite, . . . . .	-0.43	-0.55	-0.57	...
Electric Light Carbon, .	-0.46	-0.52 (?)	-0.57	...
Blende, . . . . .	-0.48	-0.52	-0.55	...
Boulangerite, . . . . .	-0.50	-0.55	-0.55	...
Bournonite, . . . . .	-0.50	-0.55	-0.56	...
Coke, . . . . .	-0.52	-0.52	-0.42 (?)	...
Ruby Silver-ore, . . . .	-0.54	-0.53 (?)	-0.54	...
Stephanite, . . . . .	-0.54	-0.55	-0.52	...
Stibnite, . . . . .	-0.56	-0.56	-0.56	...

matter of fact, the opposing E.M.F. due to the precipitated gold is infinitely small at the first moment of precipitation from the solution, and does not develop an E.M.F. of 0.218 until the gold is sufficiently thick to take a massive form, as we shall see later. Only a very small E.M.F. is therefore necessary

to begin the precipitation. When an appreciable quantity of gold has been deposited we have a zinc-gold couple formed which gives a high E.M.F., and would ultimately become, as at *d*, 0.582 volt.

A fault in using copper as a cathode surface is that, as soon as the gold begins to deposit, the copper becomes an anode like the zinc, and thereby reduces the cathode surface to that of the deposited gold. Consequently the chances of the gold molecules striking the permanent cathode surface are enormously reduced. If, on the other hand, we use couples *b* (Zn-Ag) or *c* (Zn-Pb) we get an E.M.F. at first greater than the Zn-Au couple, but ultimately equal to it. In this case the Ag and Pb, being negative to gold, never become an anode, but its whole surface is retained to receive the gold deposit. The deposited gold, however, becomes an anode, while the Ag or Pb is the cathode, but as the difference of potential between gold and Ag or gold and Pb is very small, the amount of gold redissolved in this case is inappreciable.

On examining Von Oettingen's and Christy's tables it may be observed, that the weaker the solution the lower is the potential for each metal. Thus, gold has a plus value for strong solutions and a minus value for weak solutions. From this it may be inferred that if we had two solutions, each containing the same percentage of gold, but one strong in KCy and the other weak, it would be more difficult to precipitate the gold from the strong solution than from the weak one. This is very noticeable when the current is applied from an external source, but when we are depending on generating the current by immersing zinc in the solution the results are not so apparent. This is due partly to a fall in the potential of the zinc in weak solutions, as seen in the tables, but mainly to the physical condition of the deposited metal, which is a matter that will be dealt with later.

## CHAPTER XVIII.

### PRECIPITATION OF THE GOLD AND SILVER.

#### *Section II.*

**Electro-chemical Changes.**—When simple salts of gold and silver are dissolved in water their molecules become broken down, either wholly or partly, into ions. Thus the molecules of  $\text{AuCl}_3$  dissociate into  $\text{Au}$  and  $3\text{Cl}$ , and these are free to move in all directions. If, now, an electric current is caused to pass through this solution it directs the ions, the chlorine going to the anode (where the current enters the solution) and the gold to the cathode (where the current leaves the solution).

When, however, complex salts of gold and silver, such as double cyanides, are dissolved in water, their molecules are also broken down, either wholly or partly, but in this case some complex ions are formed, of which the gold or silver is part. Daniell and Miller\* investigated the nature of the ions, and found that when an electric current is passed through a solution of  $\text{KAuCy}_2$ , or  $\text{KAgCy}_2$ , the  $\text{K}$  is the positive ion and goes to the cathode, while the negative ion consists of gold or silver and cyanogen, and this goes to the anode. Hittorf has since confirmed this, and assumes the negative ion to be  $\text{AuCy}_2$  for gold and  $\text{AgCy}_2$  for silver. That is to say, when these complex salts are electrolysed, the gold and silver move in the opposite direction to that taken in the case of the simple salts.

**Electrolysis.**—Faraday discovered early last century that all ions carry the same amount of electricity, or a simple multiple of this quantity. Thus an univalent ion carries unit quantity, a bivalent ion twice that quantity, a trivalent three times, and so on. The combining power of ions is found to be conditioned by the number of units of electricity they carry. Those which carry one unit have unit-combining power, and those which carry two or three units have twice or thrice the unit-combining power.

Metals separate from solutions in proportion to their chemical equivalents or combining weights. The actual weight of each metal that separates per unit of current has been determined with great accuracy, and these are known as the “electro-chemical equivalents” of the metals. They are the quantities of metal in grammes that precipitate when a current of one ampere flows for one second. Table XXX. gives a few of the more important metals, with their valency and electro-chemical equivalents in cyanide solutions.

\* *Phil. Trans.*, 1844, p. 1.

To determine the maximum weight of metal capable of being deposited from a cyanide solution—Let

$A$  = current in amperes

$T$  = the time in seconds

$x$  = the electro-chemical equivalent

$W$  = weight precipitated

then

$W = ATx$  in grammes

or

$W = \frac{ATx}{31}$  in ounces troy.

The above gives the maximum quantity the current is capable of precipitating, which is only obtainable with specially constructed apparatus, and from concentrated solutions of the salts. When the solution is very dilute, as in the cyanide process, the efficiency of the current falls off to a mere fraction of this

TABLE XXX.

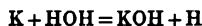
Metal.	Valency.	Electro-chemical Equivalents.	Metal.	Valency.	Electro-chemical Equivalents.
Hydrogen, . . .	H <sup>1</sup>	·000010384	Silver, . . .	Ag <sup>1</sup>	·00111800
Potassium, . . .	K <sup>1</sup>	·00040539	Copper, . . .	Cu <sup>2</sup>	·00032709
Sodium, . . .	Na <sup>1</sup>	·00023873	Mercury, . . .	Hg <sup>2</sup>	·00103740
Gold, . . .	Au <sup>1</sup>	·00203733	Zinc, . . .	Zn <sup>2</sup>	·00033696

quantity. However, when we know the number of amperes of current used, the time, and the actual weight of metal precipitated, we are readily able to compare the efficiency of the current by the above formula for any particular case.

**Electrolysis of Simple Salts.**—When simple salts, such as  $\text{AuCl}_3$ , are decomposed by the current, the Au ions give up their positive charges at the cathode and separate as atoms of metal, while the  $3\text{Cl}$  ions give up their negative charges and become electrically neutral, i.e. form free chlorine gas. Part of the free chlorine may escape to the air, but part will decompose water at the anode, forming hydrochloric acid and oxygen.

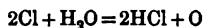
If the salt were a highly oxidisable metal, as potassium or sodium, the K or Na ions give up their positive charges to the cathode, just as gold does, and separate as atoms or molecules. Now, the potassium, in contact with the cathode, has a very high solution pressure in presence of water, and dissolves as if it were a local anode, while its cathode is the same as that of the main current. The result is that the H ions of the water give up their positive charges at the cathode, while the OH ions give up their negative charges to the potassium, or take up positive charges from the potassium, and become electrically neutral KOH. The electrically neutral hydrogen deposited at the cathode may become occluded to the point of saturation, or in other words the

cathode becomes polarised, and so long as the electro-motive force of the main current is kept greater than the opposing E.M.F., due to polarisation, so long will the potassium continue to separate and decompose water. Consequently the hydrogen will continue to deposit, and as it cannot be occluded beyond a certain point with a given E.M.F. it escapes as gas. The reaction may be expressed in its simplest form thus—



It must not be forgotten that there is also a primary decomposition of the water that liberates hydrogen at the cathode and oxygen at the anode, independent of the secondary action of the potassium. This occurs even with a very minute current.

The anion of the salt, whose nature depends on the acid, gives up its negative charge at the anode and becomes electrically neutral. It is there left free in the solution and decomposes the water, liberating oxygen, or may combine with a salt in the neighbourhood of the anode. If the salt, being electrolysed, is KCl, the liberated chlorine attacks water thus—



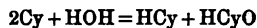
and when KOH is present we have



**Electrolysis of Complex Salts.**—It has been pointed out that when a complex salt such as  $KAuCy_2$  is electrolysed, the K ion goes to the cathode and the  $AuCy_2$  ion to the anode. It may be asked, if this is the case, how is it that the gold is ever deposited at the cathode? The only way the ion  $AuCy_2$  can get to the cathode is by a movement of the solution in an opposite direction, which overcomes the force of the current that tends to draw the gold-bearing ion away to the anode, but it is assumed by some authorities that as the osmotic pressure of the gold is very considerable, the ion breaks down again into Au and  $2Cy$  ions, and that the metal then deposits as in the case of a simple salt. It is probable that this accounts for a portion of the metal precipitated, particularly when the current is of high E.M.F.

Hittorf attributes the deposition to secondary action caused by the potassium set free at the cathode. We have followed this matter up, and from further investigation have come to the conclusion that the complex ion  $AuCy_2$  gives up its negative charge at the anode, becoming electrically neutral, and separates as gold and cyanogen atoms or molecules at that electrode.

By using an insoluble anode we are able to investigate the nature of the solution collecting around that electrode without introducing metallic ions. We find that the cyanogen liberated attacks the water and free alkali, the changes being for the most part—



and



The gold actually deposits on the anode, but in presence of Cy ions of

KCy in solution AuCy forms at that electrode, and this compound dissolves in excess of the KCy solution to form  $\text{KAuCy}_2$ , just as if a gold anode were being used.

At the same moment that the  $\text{AuCy}_2$  ion gives up its negative charge at the anode, the K ion gives up its positive charge at the cathode, and separates as an atom. The potassium is thereby made the positive electrode of a galvanic cell, its negative electrode being the cathode of the main current. The action of the potassium is then the same as that described under electrolysis of simple salts, *i.e.* local currents are continuously kept up by the separating atoms. At the same time molecules of  $\text{KAuCy}_2$  are carried, by diffusion of the liquid, into the paths of these local currents, whereby the ions are directed, the K cations giving up their charges at the cathode, and the  $\text{AuCy}_2$  anions giving up their charges at the local anode; that is, at the potassium atom separated by the main current. The gold then becomes part of the cathode of the main current, while the liberated cyanogen is left free to combine with the water or alkali in the vicinity.

A loss in cyanide always occurs when  $\text{KAuCy}_2$  is electrolysed, both at the anode and cathode.

It is not at all necessary that the  $\text{AuCy}_2$  ions should come in contact with the anode of the main current, for the reason that the useful work done by main current is in separating potassium atoms at the cathode. The secondary action caused by these atoms at the cathode, then, does the useful work in precipitating the gold, and it is only necessary that the molecules or ions containing the gold should strike the cathode of the main current where the potassium is being separated. The potassium is not necessarily derived from the  $\text{KAuCy}_2$  salt, but may be, and is in practice, derived chiefly from other salts in the solution. It is, of course, to be understood that other positive metals than potassium act as the anode of the secondary currents, and it would appear that under certain conditions hydrogen was capable of taking the part of a positive metal for this purpose.

**Diffusion at the Cathode.**—From the last paragraph it is obvious, that one of the essential points in precipitating the gold and silver from cyanide solutions, is that of rapid diffusion at the cathode of the main current.

In the cyanide process we have often to deal with a solution containing less than 1 part of gold by weight to 100,000 parts of solution, or about 5 dwts. per ton, which shows what an enormous amount of solution has to come in contact with the cathode in order to yield 1 part of gold. It is usual in extracting the gold to pass the solution between several sets of electrodes until the gold contents are reduced to 10 grains per ton, or something less than one in a million. This fact alone will emphasise the importance of diffusion.

Now, the action of the main current, as already stated, is to take the gold to its anode, and if it were not that other forces are at work, but little gold would be precipitated at the cathode. The rate at which metallic ions move by the action of the current is fortunately not great, being under 1 in. per hour with a potential gradient of 2.5 volts per inch, and this is usually over-



come to some extent, by diffusion of the liquid. In practice, the necessary diffusion is accelerated by allowing the solution to flow between the electrodes at a suitable rate.

The time required for the deposition of the gold to take place when the  $\text{KAuCy}_2$  ions come in contact with the potassium anode may be considered, for practical purposes, as *nil*, and the whole of the time of precipitation is therefore occupied in bringing these ions in contact with the electrode. The time required to exhaust a solution of its metal may therefore be decreased, by either increasing the rate of flow or by increasing the cathode surface, in such a manner that a larger proportion of molecules strike that electrode. These are practical considerations which will be dealt with later.

If it were made possible to remove the solution to another vessel immediately after it was in contact with the cathode, making room for fresh solution to be acted on, we might readily have perfect precipitation. In practice, however, the solution from which the metal had been precipitated is allowed to diffuse to the other solution still containing gold, and the result is a poorer gold solution, to be acted on in turn. Now, the poorer the solution becomes the further apart must be the molecules or ions carrying the gold, and consequently, with a constant rate of flow, the number of ions that strike the same area of cathode surface must be an ever decreasing quantity.

**Electrolysis by a Galvanic Couple.**—When a positive metal, having a difference of potential at two parts of its surface, is immersed in a salt of a less positive one, the dissolving part becomes the anode and the less soluble part the cathode, and precipitation of the less positive metal proceeds as if the current was applied from an external source. If the salt is a complex one, as  $\text{KAuCy}_2$ , the action may be said to take place as follows :—

Let us assume that the precipitating metal is zinc, which contains lead as an impurity, and the solution potassium cyanide containing some  $\text{KAuCy}_2$ . Currents are generated by the dissolving action of the zinc, as already explained, flowing through the solution from the zinc to the lead. In the paths of these currents molecules of  $\text{KAuCy}_2$  are carried by diffusion, and their ions are directed, the  $\text{AuCy}_2$  going to the zinc and the K to the lead, where they simultaneously give up their charges and become atoms or molecules. The gold deposits on the part of the zinc where the current enters ; that is, the anode and the  $\text{Cy}_2$  is left free to combine with KOH or water, as already stated. The K ion gives up its charge in the same way at the lead and becomes an atom, forming for the moment an anode, while the lead is the cathode. Gold is then deposited on the lead as the solution diffuses to that electrode, in the way described under electrolysis of complex salts. Thus, when a zinc-lead couple is employed, gold deposits both at the zinc and the lead, whereas when we apply a current from an external source, gold is deposited permanently only at the cathode. At the moment the atom of gold is deposited on the zinc its tendency is to re-dissolve, but a new positive or anode part of the zinc takes this atom as a negative electrode or cathode, on which to deposit potassium atoms. The potassium, as it separates, keeps up local currents that precipitate further quantities

of gold as the solution diffuses, and thus the deposit thickens. The proportion of gold actually precipitated by the primary action of the zinc must necessarily be small, as the surface soon becomes covered with gold, and the chief amount must be obtained by the secondary action of the potassium.

It is not necessary that the zinc should come in contact with the gold solution, provided that an electro-negative surface in electrical contact with the zinc is supplied, on which the potassium atoms may separate. Thus a surface of lead, tin, iron, etc. may be employed in contact with the gold solution, while the zinc is in a separate cyanide solution containing no gold, and separated from each other by a porous partition. All that is required then to deposit the gold, is to connect the zinc with the negative surface by a wire, and cause the gold-bearing solution to diffuse.

Neither is it necessary that the zinc should dissolve in a cyanide solution. Any other solution by which zinc is attacked will answer to a greater or less degree. In practice the solutions usually contain free alkali with the cyanide, in which case the free alkali answers the same purpose as the cyanide, that is, the production of electric currents and the deposition of the alkali metal at the negative surface.

What has been said with reference to diffusion of the solution at the cathode when an external current is applied, is of equal importance when precipitating with a galvanic couple.

**Polarisation.**—If two pieces of metal, such as lead, are placed in a salt solution, such as cyanide, and are connected with the terminals of a galvanometer and the terminals of a galvanic element, it may be observed that at the moment of contact a considerable current is indicated, while later the needle goes back almost to zero, thus showing that the current had nearly stopped. If, now, the element be cut out of the circuit, the galvanometer indicates a current almost equal to that of the maximum primary current, but in an opposite direction. This is known as the "polarisation current," and, like the primary current, will be found to fall in intensity, at first rapidly and afterwards more slowly. Polarisation may be due to several causes, but the chief one is owing to the accumulation of hydrogen at the negative electrode.

When a current is passed through an aqueous solution, hydrogen ions are always un-ionised at the cathode, that is, they separate in molecular or gaseous form, which up to a certain point, dependent on various factors, such as atmospheric pressure, temperature, etc., accumulate and become occluded. Oxygen separates at the same time at the anode, and there becomes occluded. If the primary current is cut out, we have a hydrogen-oxygen gas cell formed, short-circuited through the galvanometer, which results in a current in the opposite direction to that of the primary one. This current has an electro-motive force, dependent on the concentration of the gases, which, however, is never greater than that of the primary cell.

It was found in one case that when the concentration of hydrogen was a maximum, for a normal atmospheric pressure and a temperature of 50° F., the E.M.F. of the polarisation current was 1.1 volts. If, now, this current be

opposed by connecting the electrodes with a primary cell of 1·1 volts we have equilibrium and nothing happens, but if the primary cell be less than 1·1 volts, say 0·6 volt, we get a polarisation current of 0·5 volt ( $1·1 - 0·6$ ), which gradually diminishes until the concentration of the hydrogen is such as would produce an E.M.F. of 0·6 volt. On the other hand, if the E.M.F. of the primary cell is greater than 1·1 volts, say 1·3 volts, then water is decomposed, with a current having an effective E.M.F. of 0·2 volt, and hydrogen and oxygen escape to the air. We must therefore remember that this opposing force is always present when precipitating metals from solutions. By raising or lowering the atmospheric pressure the E.M.F. of the polarisation current is raised or lowered accordingly, and by increasing the E.M.F. of the primary current the E.M.F. of the polarisation current is also increased, but not in proportion, there being always a falling off in the latter as the former becomes higher.

## CHAPTER XIX.

### PRECIPITATION OF THE GOLD AND SILVER.

#### *Section III.*

**The Significance of E.M.F. for Precipitating.**—It is well known to analysts that in the precipitation of metals by electrolytic methods, every metallic salt is only capable of being decomposed and the metal precipitated continuously, when the E.M.F. of the current has reached a certain voltage. This varies with different metals and with different salts of the same metal. Thus the E.M.F. for a continuous precipitation of silver from normal solutions of silver nitrate has been found to be 0·7\* volt, while if a current of the same E.M.F. was applied to a normal solution of potassium silver cyanide we should find little or no silver on the electrode; and again, a current with an E.M.F. that would just precipitate silver from a cyanide solution would not be sufficiently high to precipitate zinc from the same solution.

Le Blanc investigated this subject, and found that the cause of a metal not being continuously precipitated until a definite E.M.F. is reached may be understood from the fact that when the metal in question is precipitated at the cathode it exhibits an opposing E.M.F., at first very small, which goes on increasing to a maximum, *i.e.* until the deposit is in a massive form. When the E.M.F. of the primary current is at or near that of the opposing E.M.F. of the massive metal at the cathode, we have the minimum E.M.F. of decomposition, and any E.M.F. above that effects a continuous deposit. Every metal dissolved in a solution has a certain tendency to change into the metallic state, some more than others, just as every metal has a tendency to change into the ionic state, or, as we say, to dissolve. Gold ions have a great tendency to change into the metallic state when compared to zinc ions, and when in the metallic state the gold has a much smaller tendency to change into the ionic state than zinc. From this it follows that when precipitating these metals the opposing E.M.F. of the deposited metal must be greater in the case of zinc than gold, and therefore the minimum E.M.F. of decomposition must be higher in the case of zinc than of gold.

When a piece of zinc is placed in a cyanide solution containing a few pennyweights of gold per ton, very thin films or a number of specks of gold are first deposited. The difference of potential between the gold and the zinc is then of great magnitude, but this goes on decreasing as the

\* Le Blanc, *Zeit. physik. Chem.*, viii. 299 (1891).

deposit increases in thickness, until the latter assumes the massive condition of a plate of gold. For the same reason highly oxidisable metals separate as atoms with a current density of low E.M.F., but they cannot assume the massive form until the E.M.F. of the primary current is greater than that produced by the massive metal.

The nature of the electrode on which the gold is precipitated has influence on the effective E.M.F. of the primary current. Thus the E.M.F. that would just precipitate gold on a gold cathode would not be effective in precipitating gold on a more highly negative substance, such as carbon, for the reason that the precipitated gold in contact with the carbon re-dissolves in presence of free cyanide, due to the formation of a gold-carbon couple.

**Physical Effect of the Nature of the Electrodes.**—The minimum E.M.F. of decomposition is found to depend not only on the metal and the nature of the salt, but on the physical condition of the surface of the electrodes. Thus, if the electrodes have a hard smooth surface, such as rolled metal, the decomposition point is found to be much higher than when the surface is of a granular, spongy, or slimy nature. This is very marked in the case of platinum electrodes, and is more or less marked in the case of other metals. Thus, when the electrodes consisted of hard rolled platinum foil, we found at normal atmospheric pressure and 50° F. the minimum decomposition point of water to be 1.8 volts, whereas, when the electrodes were platinised, under similar conditions, the decomposition point was only 1.1 volts. When the cathode consisted of hard rolled lead, we found that the decomposition point was always higher than when it consisted of spongy lead. Again, if a zinc plate is the positive electrode of a simple cell and rolled lead the negative electrode, the electrolyte a solution of pure  $\text{KAuCy}_2$ , little or no gold is precipitated. If next we substitute a negative electrode of spongy lead, the gold rapidly precipitates. In the first case, with rolled lead the decomposition point was higher than the E.M.F. between the lead and zinc was capable of effecting, but in the case of spongy lead the decomposition point was reached although the E.M.F. was the same as in the first case. If the lead is jagged or granular it has the same effect as spongy lead, differing only in degree. When lead is deposited in a spongy form on zinc the gold precipitates for the same reason.

—The phenomenon is traced to the occlusion of hydrogen. With smooth surface cathodes the hydrogen is only slightly occluded, and a film of the gas separates the metal from the solution. The film of gas introduces a resistance into the circuit, and to overcome this a higher E.M.F. than is available would be necessary. When, however, the surface is granular or spongy, it is capable of occluding the hydrogen and the surface film does not appear, or only partially, so that the available E.M.F. is sufficient to effect decomposition.

It is well known to those accustomed to precipitating gold with zinc shavings, that when dealing with dilute solutions the gold is apt to adhere as a thin hard film, after which the precipitating properties of the zinc seem to be

almost at an end. This appears to be due simply to the raising of the decomposition point above that which the zinc is capable of effecting. The hard film of gold has the same properties as that of hard rolled metal. Anything that will cause the surface to be roughened will lower the decomposition point. In practice, this is done by adding cyanide or alkali to the solution, so as to temporarily increase the strength of the solution. The precipitated gold is then caused to partly shell off or loosen and form a number of jagged edges, which admit hydrogen to become occluded. The addition of cyanide or alkali not only lowers the decomposition point, but slightly increases the E.M.F. between the zinc and gold, and thereby increases the current density for the time being. This causes a slimy loose deposit to form, and thus the danger of raising the decomposition point again is diminished. In practice, this danger is often obviated by the method, patented by J. S. Macarthur, of dipping the zinc into acetate of lead solution before putting it into the precipitating box. This causes a loose spongy lead deposit on the zinc, which acts as the negative electrode and forms the so-called 'zinc-lead couple.' Then when the gold solution is passed through the mass of zinc-lead couples it meets with a ready formed negative surface favourable for maintaining a low decomposition point of the solution. On the other hand, if we start with clean zinc, a considerable time must lapse before a suitable negative surface is formed, and there is always the danger of this surface forming so as to raise the decomposition point. This danger is not so great in strong solutions as in weak solutions; and if the solution is sufficiently strong in cyanide and rich in gold to produce a loose slimy deposit, there can be little or no advantage in precipitating the spongy lead on the zinc.

**Influence of Current Density.\***—If the area of the anode be great and the cathode small, we may get, even when the current strength is small, a high density current at the cathode; but when the current is reversed, it would give a comparatively low current density at the cathode. This is illustrated graphically in fig. 24. If N and M represent the edges of circular discs in a solution of a salt, and a current be sent from N to M through the solution, the current becomes more dense at M than at N, as shown by the converging lines; but if the same current be sent from M to N, the current becomes less dense at N than at M, as shown by the diverging lines. Now, in practice we find that with a low density current a hard smooth film of metal adheres firmly to the electrode, as for instance when a metal is precipitated at N, but if the density is high as at M, the deposit becomes of a granular or loose slimy nature.

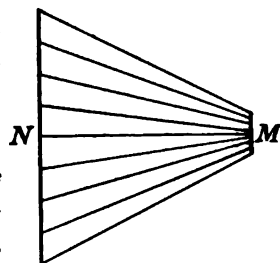


FIG. 24.—Low and high current density.

\* By current density is meant the ratio of the strength of the current passing, to the area of the anode or cathode. Thus, if A is the strength of the current in amperes, and B the area of the electrode in feet, then  $\frac{A}{B}$  = density of current per square foot.

These are physical effects due to current density. In precipitating a metal by a current applied from an external source, as a dynamo, the physical nature is often a matter of no consequence, but there are special cases where it is a matter of great importance. When gold and other metals present are precipitated as a hard film, considerable difficulty has been experienced in removing the gold without destroying the cathodes. Chas. Butters, however, got over this difficulty by employing a high density current, which deposits the metal in such a loose form that it could be wiped off the electrode as slime or powder.

If the current is obtained from an internal source, as that obtained by the dissolving action of the solution on zinc, the current density is a matter of much importance. For if the current density be low, the gold precipitates as a hard film, which has the physical property of raising the decomposition point of the solution, as already explained, to such an extent that the E.M.F. between the zinc and gold is not high enough to effect precipitation.

On the other hand, if the current density be high, a loose black slimy deposit results, which causes the decomposition point to remain low, and then the E.M.F. of the current between the gold and the zinc is sufficient to effect continued precipitation.

In the cyanide process we have often to deal with a solution containing only a few pennyweights of gold or silver per ton, in presence of some pounds weight of free cyanide per ton, that is all the time re-dissolving a portion of the metal as deposited. If the cathode surface is small, a given current strength may produce a thick deposit, and as only a small amount of the metal is exposed to the solution, but little is re-dissolved. When the cathode surface is larger, a thinner film of metal is deposited in the same time, and as it exposes a proportionally larger area to the solution, more of the precipitated metal is re-dissolved. An extreme case would be where the cathode was so large, for the current strength, that the metal is re-dissolved as fast as it is deposited.

When employing a current from an external source, such a state of things as the latter ought never to be possible, as we are able to measure and control the current strength, but when precipitating with zinc, we may have, unobserved, an excessively large gold surface and a very small zinc surface; consequently the current density would be low, and would allow a large portion of the precipitated metal to be re-dissolved.

It must not be inferred from what has been said that the smaller the cathode the better the precipitation, for it is obvious that the smaller the cathode the fewer will be the number of molecules that strike the surface in a given time with the same rate of diffusion. It is evident, therefore, that for every given set of conditions, there must be a certain sized cathode that will give a maximum efficiency of precipitation. This may be determined in an empirical way when we use a current from an external source, but when precipitating with zinc, we can only take the precaution of keeping a preponderance of zinc surface exposed to the solution, and at the same time as large a negative surface as practicable.

**The Effect of Mixed Salts on Precipitation.**—Cyanide solutions that have been in use for dissolving gold and silver from ores usually contain impurities dissolved out of the ore. These to some extent accumulate, until we find foreign substances present ranging from a trace to 1 or 2 per cent., while the total amount of KCy present may be under 0.3 per cent. As might be expected, these impurities have an influence on precipitation.

When a current is passed through a mixture of salt solutions, it is generally found that the less oxidisable metals deposit with the least E.M.F., but the ions of all the salts migrate towards their respective electrodes. If the E.M.F. is increased sufficiently to deposit the more oxidisable metals, then they all deposit together, and if the more oxidisable metals are capable of decomposing a metallic salt or water, secondary action occurs which also precipitates the less oxidisable metals, when their molecules get by diffusion in contact with the cathode. If the solution contains, say,  $H_2O$ , KCy, and KCl, then the K of both salts and the H of the water goes to the cathode, where, by secondary action, they precipitate the less oxidisable metals that may be in contact with the electrode at that moment, or the energy may be simply used in decomposing water. At the anode O, Cy and Cl are separated simultaneously with the K and H. These not only act on the water, as already stated, but act on each other. It is easy to see that in mixed solutions if a number of anions separate at the same time, they may combine with each other, or with compounds in the solution, to produce many complexes. In practice, however, the solutions being very dilute, it is improbable that the anions, as they are liberated, combine with each other to any extent, but for the most part decompose water.

When precipitating with zinc, some of the impurities introduced into the solution have a detrimental effect. The worst of these are reducing agents, such as sulphides and organic matter. Reducing agents, generally, retard precipitation of the gold and silver, by lowering the E.M.F. of the current and hindering depolarisation of the negative surface, and consequently lessen the quantity of current generated. Soluble sulphides act on the zinc, forming an insoluble coating that renders it less soluble. With strong cyanide solutions the effect of sulphides is less than with weak solutions.

We have made some determinations of the effect of minute quantities of  $Na_2S$  in a KCy solution of 0.25 per cent. on the precipitating properties of zinc. These are plotted in fig. 25.

Cyanide solutions often become acid from being in contact with acid ore when the precipitation of gold and silver is affected to a marked degree.

**Precipitation from Acid Solutions.**—When a solution becomes acid its dissolving action on gold and silver almost ceases, but any gold or silver that had been dissolved while in an alkaline condition still remains in solution. This is supposed to exist as hydroaurocyanic acid,  $HAuCy_2$ . When a current is passed through this solution between insoluble electrodes in either an acid or neutral state, the gold and silver are readily precipitated at both the anode and cathode. That at the anode becomes  $AuCy$  or  $AgCy$  by the action of



free  $\text{HCy}$  in the solution, and this compound retards the current when the solution is rich in metal, but in highly attenuated solutions the  $\text{AuCy}$  or  $\text{AgCy}$  dissolves in the excess of  $\text{HCy}$ . The gold or silver precipitated at the cathode remains permanent. The potential of both metals with respect to acid solutions is a minus quantity, and therefore there is little or no tendency to re-dissolve.

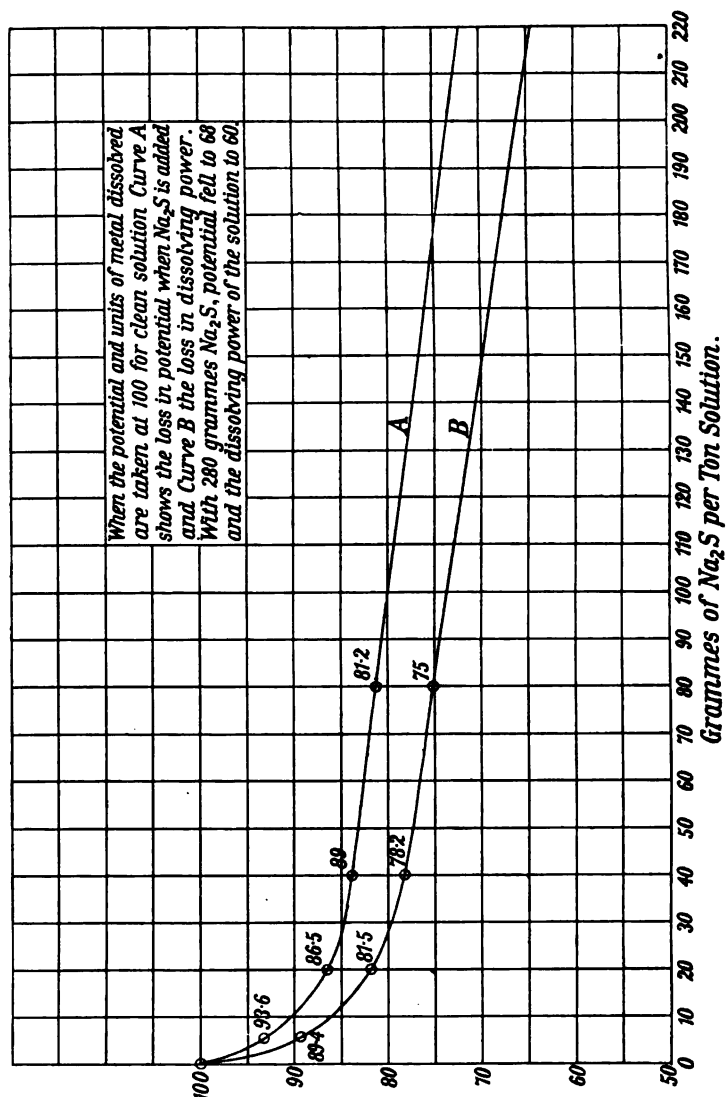


FIG. 25.—The effect of Sodium Sulphide on the precipitating properties of Zinc in a 0.25 per cent.  $\text{KCy}$  Solution.

Neither zinc nor aluminium precipitate an appreciable quantity of gold from  $\text{HAuCy}_2$  solution if neutral, but if it contains an excess of a strong acid ( $\text{H}_2\text{SO}_4$ ), precipitation is fairly rapid and perfect. For obvious reasons the use of acidified solutions is impracticable.

**The Velocity or Rate of Flow past Electrodes.**—It has been pointed out that by increasing the rate of flow through the precipitating box, we bring a larger number of  $\text{KAuCy}_2$  molecules in contact with the electrodes in the same time. This increase in rate of flow has also another advantage.

The cathode always becomes polarised with hydrogen, which causes the E.M.F. of the primary current to be reduced, and if the E.M.F. of the primary current is small, it may be almost entirely stopped by the opposing E.M.F. of the deposited hydrogen. Now, it is well known that the solution in contact with air absorbs oxygen; and we find that if this oxygenated solution be brought into contact with the polarised cathode, depolarisation is effected and the primary current regains its original E.M.F., the reason being that the occluded hydrogen combines with the absorbed oxygen in the solution, forming  $\text{H}_2\text{O}_2$  and water.

If the solution is motionless this regain of E.M.F. is only momentary, because the oxygen molecules in contact with the electrode become used up, and new hydrogen atoms are instantly deposited to take the place of those that combined with the oxygen. But when the solution is caused to flow past the polarised electrode, so as to produce a continued supply of oxygen molecules, we get a continued depolarisation, and if the rate of flow is sufficiently rapid, the hydrogen will be removed as fast as it is deposited.

In a specially constructed experimental precipitating box to determine the effect of the rate of flow on polarisation, we found that when the solution had been standing still some hours the E.M.F. of the current between precipitated gold and zinc was 0.01 volt. The head of the box was then elevated to different heights and the solution was run through at the greatest capacity of the box. When the slope was 1 in 30 the voltage was 0.08, when 1 in 20 it rose to 0.23, 1 in 15 to 0.33, 1 in 13 to 0.42, and 1 in 12 to 0.56, and with a greater velocity in the rate of flow the E.M.F. remained constant. The current was generated in the above box by dissolving zinc in a solution of  $\text{KC}_2\text{O}_8$  and  $\text{NaOH}$ .

When the current is supplied from an external source this effect of polarisation is not a matter of great importance, as it is easy to increase the E.M.F. so as to overcome the opposing E.M.F. of polarisation. But in precipitating with zinc we are restricted to the E.M.F. between the zinc and the gold, which in working solutions is seldom greater than 0.55 volt. The specific rate of flow of a solution through a zinc precipitating box is therefore a matter of practical importance, other than that of diffusion of the  $\text{KAuCy}_2$  molecules already referred to. For, consider two boxes each containing the same quantity of zinc, but in the first the zinc column had twice the length and half the sectional area of the other. Then for equal quantities of solution run through the rate of flow would be twice as fast in the first as in the second, but it would have twice the length of column to traverse, and therefore the time of contact with the zinc would be the same in either case. If the precipitation was simply a matter of time of contact with the solution it should be as good in the first as in the second case, but practical experiment is against this, and the reason may be surmised from the above results.

## CHAPTER XX.

### PRECIPITATION BY ZINC.

THIS was the first commercially successful method of precipitating gold and silver from the solutions. The credit of its success is due to Macarthur and Forrest, who prepared the zinc in the form of fine thin shavings or turnings. Previously, zinc plates had been used or proposed, but without success, and it was not until 1888, when these inventors conceived the idea of cutting the zinc into fine ribbon-like shavings, that any practical method of precipitation had been put forward, and for this reason, if no other, precipitation by zinc will always possess a scientific and historical interest.

As is well known, the method consists simply in passing the gold solutions through a mass of zinc shavings loosely pressed together, when the gold and silver deposits on the zinc surface. This is usually done in a long trough-shaped box (see figs. 106 to 110), divided into a number of compartments, so arranged that the solution flows upwards through alternate wide divisions containing the zinc, and downwards through alternate narrow divisions containing only the solution. In some cases the box has been made with all divisions of the same width, and all containing zinc, so that the solution flows down through one compartment and upwards through the following, but the former plan is generally adopted. The zinc rests on loose false bottoms or trays covered with sieving or perforated plate, which are arranged at a height of 4 to 6 inches from the bottom of the box,—the object being to leave a clear space where the precipitate can collect without being disturbed, and, further, to facilitate an even flow of the solution.

**Preparation of Zinc Shavings.**—These were originally prepared from rolled sheet zinc cut into discs of 5 to 12 inches diameter, which had about 1-inch holes punched in their centres, and about a dozen at a time were clamped together on a screwed mandril. The mandril was secured to a simple lathe headstock fitted with cone pulleys, and the shavings were turned off with a broad carpenter's chisel. The chisel was supported on a hand rest and held obliquely, so that the discs as turned took a somewhat conical shape. The width and thickness of the shavings could be varied at will, according to the wish of the operator. The best results are usually obtained at a speed of about 200 to 250 feet per minute, but this may be made to vary rather widely with the shape of the turning tool.

The above method is still largely used where labour is cheap, but where wages are high it is preferable to use one of the many specially designed lathes for cutting shavings with an automatic self-acting slide rest. Amongst these

are Liddell's lathe, as supplied by the Cyanide Plant Supply Company, which is said to be capable of cutting shavings from discs  $\frac{1}{800}$  of an inch thick at the rate of 150 lbs. per day of ten hours.

Another special form of lathe, known as the "Hampton," has recently been introduced in America by the Pacific Tank Company, for producing shavings from rectangular sheets without previously cutting into discs. This machine has a mandril with a keyway into which the edge of the sheet is inserted and fixed by a key. The sheet is then wound on the mandril, a second sheet is lapped under the end of the first and wound on also. The roll of metal is tied together by wire twisted round it, and the shavings are turned off the end of the roll by a square-nosed cutting tool, actuated by an automatic feed of the usual kind. A roller attachment is provided in the latest patterns for leading off the shavings from the cutting tool, and thereby reducing the necessity of superintendence to a minimum.

In New Zealand and the United States very narrow filaments are preferred, but in many of the South African works a fairly broad shaving is found most serviceable. Theoretically, the thinner the filament the better, as the surface is proportionally larger for the same weight, but in practice it is found that when very thin the shavings break up into short lengths, agglomerate, and choke the zinc box, resulting in an uneven flow and an excessive quantity of finely divided zinc mixed with the gold slimes.

On the other hand, if too thick the surface may be much reduced for the same weight of zinc, and when in use for some time lose, to some extent, the precipitating property of the thin shaving. A suitable thickness is found in practice to be  $\frac{1}{160}$  to  $\frac{1}{800}$  of an inch, and some operators prefer to mix shavings of different thicknesses. The width in practice varies from  $\frac{1}{32}$  to  $\frac{1}{4}$  of an inch, a  $\frac{1}{8}$  to  $\frac{1}{2}$  being preferred in most cases.

The importance of the width and thickness of shavings, however, largely vanishes when we employ a zinc-lead couple, described elsewhere.

**Minimum Strength of Solution.**—Much diversity of opinion exists as to the weakest solution that can be used for efficient precipitation. This follows from the fact that, under working conditions, there exist a number of variable factors which are not taken into account, and thus make comparisons of little value. All authorities are, however, of the opinion that strong solutions give uniformly good results, while weak solutions give results that are more or less erratic.

John Yates \* says, in 1896, on the Rand, with solutions of 0.2 per cent. to mere traces of KCy, it was not uncommon to have the gold solutions in the sumps down to  $\frac{1}{2}$  dwt., and he instances cases where a few grains were frequently found. He, however, thinks that in slimes treatment when the strong solution is 0.008 per cent. and washings proportionately weak, zinc does not give satisfaction. L. S. Bosqui † found that with 0.14 to 0.16 per cent. solutions he reduced 3 dwts. 21 grs. down to 2 grains per ton, and with 0.04 to 0.07 per

\* *Jour. Chem. and Met. Soc. S. Africa*, vol. i. p. 257.

† *Trans. Am. Inst. Min. Eng.*, xxvii. p. 837 *et seq.*

cent. solutions from 2 dwts. 7 grs. down to 2·36 grains per ton. Alfred James\* says, that in New Zealand "solutions containing less than 0·05 per cent. KCy are regularly reduced to below one grain of gold per ton, whilst in Mexico and the United States almost equally satisfactory precipitations are not uncommon." He further adds that with very foul solutions containing iron and alumina salts, but *no* free cyanide, one may in practice reduce the bullion contents to 1 or 2 grains per ton, by allowing the solution to flow through the boxes at the rate of half a ton per cubic foot of zinc per day.

From James' testimony it would appear that when precipitating from weak solutions, or those containing no free cyanide, it would only be necessary to increase the capacity of the zinc boxes when the presence of cyanide becomes unnecessary. On the other hand, according to Yates, 0·008 per cent. KCy is below the limit at which satisfactory results can be obtained.

In spite of the above and similar conclusions by various other authorities cyanide men are constantly meeting with difficulties, in getting satisfactory precipitation from very weak solutions by means of zinc, the results being at one time good, and at another time bad, without any apparent reason.

On investigating cases similar to those referred to by James, we have found that a satisfactory precipitation was obtained in very foul solution without diminishing the rate of flow, even in the absence of a trace of free cyanide, while with clean solutions containing as much as 0·06 per cent. free cyanide the extractions were invariably unsatisfactory. In seeking for the reason of this, we find that the amount of free cyanide present in solutions is only one of several factors that cause precipitation; and further, that when the solutions contain very little free cyanide in presence of larger quantities of other salts, the amount of free cyanide is a factor that practically vanishes altogether. The chief factors of importance are the difference of potential between the zinc and the gold with respect to the solution, the current generated by the dissolving of the zinc, and the physical condition of the negative surface which receives the deposit. Assuming that, in all cases compared, the latter factor is always the same, the question resolves itself into one of potential difference and current strength. In very weak solutions of pure cyanide the potential of zinc falls considerably, as shown by Von Oettingen and Christy, Tables XXVIII. and XXIX., pages 111 and 112, and it may be assumed that when no cyanide was present it would be near zero, and no precipitation could take place. But in working cyanide solutions that have been in use for some time, and especially in the so-called foul solutions, the potential difference is practically the same with a little KCy as without. Therefore, as far as this factor is concerned, the presence of a little cyanide in foul solutions has little or no effect. Then again, in foul solutions containing no cyanide, we find that the current strength is often practically unaltered by the addition of a small quantity of KCy, such as is found in the more dilute solutions,—the reason being that the zinc is acted on by other salts present, which often amount to

\* *Trans. Am. Inst. Min. Eng.*, xxvii. p. 278 *et seq.*

1 per cent., while the addition of such a small quantity of KCy as 0·01 per cent. has little or no effect.

On the other hand, if we take a solution that is not foul and add to it a little KCy, it produces a sensible effect both on the potential difference and the current strength, for then the KCy present becomes a factor of importance; but other salts would answer, more or less, the same purpose, as already stated. Therefore we cannot rely on the amount of KCy present in these dilute solutions as any guide for good precipitation.

**Zinc-lead Couple in Practice.**—We have already referred to the physical condition of the negative surface as one of the factors that affects precipitation. In strong cyanide solutions or solutions containing a large percentage of other salts this factor is not very apparent, for the reason that the gold deposits more or less, so as to produce a negative surface of the requisite physical condition, but when the solution is very weak in salts, the gold deposits as a hard film, which raises the decomposition point, as already described.

W. K. Betty observed that when a zinc-lead couple was used with very weak cyanide solutions, even in absence of any appreciable quantity of other salts, as is the case in the treatment of slimes, a satisfactory precipitation could be obtained. The use of this couple had been previously patented by Macarthur, but when it was applied to the strong solutions then used in sand treatment, its value was not appreciated, because the gold deposited in a loose powdery form, and therefore the zinc-lead couple was uncalled for.

The method of preparing the zinc-lead couple is by immersing the ordinary zinc shavings in a solution of lead acetate containing from 1 to 5 per cent. of lead, just before being placed in the precipitating box. By this treatment metallic lead is precipitated in a loose spongy form on the zinc to the extent of about 1 per cent. of the weight of the latter.

The boxes used are usually the same as for the older process, and the rate of flow, with very dilute solutions, varies from  $\frac{1}{2}$  to 1 ton of solution in twenty-four hours for each cubic foot of space occupied by the zinc, the specific rate of flow being from 0·11 to 0·33 ft. per minute. With this rate of flow W. A. Caldercott gives the following results from six of his circular precipitation vats, working in pairs, at the Witwatersrand Gold Mining Company.\*

Solution entering first vat, 2 dwts. 20 gra. per ton.			
„	leaving	„	10 „ „
„	„	second vat,	5 „ „
„	„	third „	4 „ „

These vats were each 5 ft. in diameter and 3 ft. deep inside.

The following figures published by A. H. Hartley† relating to the zinc-lead couple are interesting, because some copper in addition to the gold was present in the solution. From these four metals six different couples might be formed,

\* *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. No. 11.

† *Ibid.*, vol. ii. p. 101.

which would render the action somewhat complicated ; nevertheless the results were satisfactory.

Per cent. KCy.	Gold at head of box.	Gold at foot of box.
0.02	... 1 dwt.	... trace
0.007	... 20 dwts.	... "
0.004	... 2½ "	... "

When the zinc-lead couple was first applied to very dilute solutions, great stress was laid upon the necessity of adding KCy at the head of the boxes for a few hours, each time they were filled with freshly prepared zinc, but since the general adoption of the process it has been found that this precaution is unnecessary. The ordinary practice is to put newly prepared zinc in the boxes which receive the weakest solutions, and to transfer it later to the boxes for strong solution.

**Consumption of Zinc.**—The actual amount of zinc dissolved during the precipitation is a matter that is not easy to determine, but it is certainly many times in excess of the theoretical quantity required. If the whole of the zinc dissolved was used to precipitate the gold or silver in the solution, 1 ounce should deposit 6 of gold or 3 of silver, and this is about what we should get in a concentrated solution of  $\text{KAuCy}_2$  or  $\text{KAgCy}_2$ , for then the molecules of the salt are so numerous that as one is decomposed and the metal precipitated, fresh molecules instantly come in contact with the metallic surface and are also decomposed as the zinc dissolves. In the cyanide process, however, we have to deal with solutions containing about 1 part of gold in 100,000 to 1,000,000 parts of water, so that the molecules of the salt are so far apart, as to make it impracticable to bring the gold-bearing molecules in contact with the metallic surface, as fast as required by the dissolving zinc. The result is that we get a hundred or more zinc molecules dissolved for every molecule of the gold salt that strikes this surface at the right spot and at the right time. The zinc is therefore for the most part wasted in decomposing water and liberating hydrogen.

It follows that the richer the solution is in gold, the smaller will be the amount of zinc required to precipitate the same weight of metal, provided the solutions are alike in strength as to other salts. Strong salt solutions dissolve a larger quantity of zinc in the same time than weak solutions, but in practice the weak solutions are poorer in gold than the strong ones, so that the amount of zinc dissolved per ounce of gold precipitated in either case does not vary as much as might be expected.

**Zinc Box Operations.**—In charging a zinc box it is preferable to select a portion of the shavings somewhat coarsely cut, and to lay these on the trays in each compartment to a depth of a few inches. On the top of these the thinner shavings are placed by drawing them loosely asunder and spreading as evenly as possible, taking care to press them gently into the corners to avoid any channels of easy flow. The last compartment is generally left empty to act as a settler, which is to avoid the danger of the precipitate being carried away in suspension in the solution. This compartment has

sometimes a screen of coarse hessian or burslap placed near the surface of the solution, in order to act as a filter and check any fine slime being carried away. Its use is, however, seldom found of value.

In re-charging the zinc box the old shavings are usually placed in the compartments at its head, and those at the foot are re-charged with new material. It is good practice always to place a little of the new coarsely cut shavings on each tray before putting back the old shavings.

The gold-bearing solution is allowed to flow through the box at as uniform a rate as practicable. The most suitable rate of flow, which varies with the sectional area of the column of zinc, with the area of the zinc surface, and other factors, is found by trial.

With a little practical experience the operator gets a rough general idea as to whether precipitation is effective or not. If the precipitate is loose, of a black or dark-brownish colour, it may be taken for granted that the gold is precipitating satisfactorily. If, on the other hand, the deposit is firm and of a lightish colour, it is probable that the sump solutions will have high assay values. It is usual then to take the precaution of adding a few pounds of cyanide lumps or caustic alkali to the head of the zinc box, and thus temporarily increase the action on the zinc. The lightish colour then disappears and the deposit should become loose. The light-coloured precipitate is not always a sign of poor precipitation, as it is often caused by aluminous or other matter precipitated from the solution, but it is generally advisable to keep any such precipitates at a minimum. If this cannot be done by temporarily increasing the strength of the solution, it then becomes advisable to clean-up often.

After the solution has been flowing some time the zinc is apt to become loose and to float, due to a portion dissolving and to a large amount of adhering hydrogen. When this happens, channels of easy flow are apt to form around the sides and in the corners of the compartment, resulting in a poor precipitation. This is prevented by a careful examination of the boxes at least twice a day, and a little fresh zinc added, particularly at the corners, when an appearance of looseness is observed. When any movements of the zinc are made, or when fresh zinc is added, it is usual to stop the solution for a time to allow loose precipitate to settle.

**Copper in the Solutions.**—Much difficulty has often been experienced by the presence of copper in the solution. Zinc in a cyanide solution precipitates copper, but the potential difference between the two metals is small compared with that between zinc and gold, or zinc and silver. The effect of this is that a solution which would just precipitate a loose black deposit of gold would not be at all suitable for producing a loose deposit of copper. If copper were present in such a solution it would be precipitated as a firm metallic coating, such as would raise the decomposition point of the solution above that which the potential of the zinc was capable of effecting, and then the action falls off. Should the solution be sufficiently strong in salts to deposit the copper in a loose form, both gold and copper precipitate together ;



but as the solution pressure of copper is higher than that of gold, a larger proportion of the former is re-dissolved, and the net result is a poorer precipitation.

The strength of the solution required to produce this loose deposit of copper is not often met with at the present day in treating gold ores, and it is therefore advisable when copper is troublesome, from an economic point of view, to resort to other methods of precipitation, such as the electrical process described later. When copper is known to be present in an ore, care should be taken to make exhaustive quantitative tests before deciding to employ zinc as a precipitant.

**Efficient Precipitation.**—Good and bad precipitation is largely a matter of opinion. Thus, in one work where the sump solutions never exceed 1 dw't. the precipitation is regarded as efficient, whereas at another work 10 grains is considered the maximum for efficient precipitation. There are some who reduce their solutions to 1 or 2 grains, but this is often a doubtful economy.

**Data for Designing Zinc Boxes.**—The following data taken from practice may be useful in designing boxes, and for purposes of comparison. They represent the average results of assays and measurements for twenty consecutive days, during which period 4746 tons of solution passed through the boxes.

Average value of solution before precipitation, 18 grs.

Average value of solution after precipitation, 1·5 grs.

Daily tonnage of solution, 237·3.

Gross capacity of boxes, 454 cubic feet.

Cubic feet in boxes for each ton treated, 1·91.

Space actually occupied by zinc-lead, 240 cu. ft.

Solution per day for each cubic foot of zinc, 0·99 ton.

Longitudinal section of zinc space, 16 sq. ft.

Length of zinc space, 15 ft.

Rate of flow, 5·27 ft. per minute.

Specific rate of flow, 0·33 ft. per minute for each square foot of section.

#### PRECIPITATION BY ZINC FUME.

This method was introduced by H. L. Sulman in 1894. It has been fully described by him \* and by Hugh K. Picard,† who successfully carried out the process at Deloro in Canada. The following description is principally compiled from these two sources.

Zinc fume consists of a fine powder of metallic zinc and some impurities. According to an analysis by R. H. Harland, the latter are chiefly, lead 1·74 per cent., cadmium 0·69 per cent., iron 0·11 per cent., arsenic trace, siliceous matter 0·19 per cent., carbon 0·52 per cent., and zinc oxide from 3 to 5 per cent. The latter is removed by treating the fume with weak ammonia, after which the fume is ready for use.

The apparatus used is an inverted cone made of  $\frac{3}{8}$  inch steel, usually about

\* *Jour. Soc. Chem. Ind.*, xvi. p. 961, 1897.

† *Trans. Fed. Inst. Min. Eng.*, xv. p. 417, 1897-8.

5 feet diameter and 5 feet deep, coated on the inside with zinc fume in boiled oil or galvanised. This size is capable of treating about 2 or 3 tons of solution per hour. At the bottom of the cone is a three-way cock for the admission of the liquor to be treated, and to discharge the precipitate. Immediately above this is a perforated conical rose, intended to split up the inflowing liquor, and above this is a central pipe, spread out to a funnel shape at each end. The zinc is fed into the top funnel, and meeting with the rising solution, a complete mixture is effected.

Around the top of the cone is a circular launder to receive the overflowing liquor, from whence it passes to a settler, and on to storage vats, to be used again. The settler is 8 feet long by 1 ft. 6 in. wide and 2 ft. deep. It contains a number of transverse baffle plates of glass or smooth wood, set at an angle of 45°, with half-inch spaces between them. The liquor enters at the bottom, and as it rises slowly through the baffle plates, deposits particles of fume or gold on them which may have escaped from the cone.

The following quantities were used for 15 tons of strong solution carrying 6 oza. per ton. When the liquor supply is started, 12 lbs. of zinc fume are put into the bottom of the cone, and after 3 tons have passed, 1½ lbs. of fume made into an emulsion are fed into the central tube. After 6 tons have passed, another 1½ lbs. are added, and after 9 tons, 2 lbs. more, and a further 2 lbs. after 12 tons have passed, making a total of 19 lbs. for the 15 tons. Picard recommends a rate of flow of 2 tons per hour for 6 oz. liquor.

For 14 tons of weak solution, 17 lbs. of fume were used, which was added more frequently than with strong solution. The total consumption is therefore 36 lbs. of fume for 29 tons of liquor, or 1·24 lbs. per ton solution. About 2 dwts. of bullion were left in the solution per ton. This could have been lowered by a larger consumption of zinc. The precipitate was found to contain from 8 to 30 per cent. of bullion.

Another method of using zinc fume practised at De La Mar's mines, Mercur, Utah, has been described by G. A. Packard.\*

For the daily treatment of 500 tons of ore, there were three precipitating tanks, each 14 ft. diameter and 8 ft. deep. In these the zinc dust was added to the solution, which was then agitated by compressed air forced in at the bottom of the tank. The treated solution was next forced through filter presses, five being in use, having each 18 frames 2 ft. square. Sometimes the precipitate was allowed to settle before drawing off the solution, and at De La Mar's mill the outlet pipe was 8 inches above the bottom of the tank, leaving much precipitate to be cleaned up by hand. It is said that by this method the bullion recovery fully equals the theoretical extraction, and that 0·2 to 0·25 per cent. KCy solutions can be reduced to 5 grains of gold per ton, weaker solutions (0·05 per cent.) to 2 grains per ton, with a consumption of about ¼ lb. zinc per ton of ore.

\* *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. p. 128, 1899.

## CHAPTER XXI.

### ELECTRICAL PRECIPITATION.

**General Considerations.**—In all precipitation processes now in general use electro-chemical action takes place, although the term 'electrical precipitation' is usually limited to those methods in which the current is supplied from an external source, such as from a dynamo, or primary or secondary cells.

As already pointed out, when a current is applied to a cyanide solution containing gold or silver, the K ions travel towards the cathode and the  $\text{AuCy}_2$  or  $\text{AgCy}_2$  towards the anode, and also it has been shown how important it is to overbalance this influence of the current by giving motion to the solution, so as to bring the gold- and silver-bearing ions in contact with the cathode. Some, however, do get to the anode, where they give up their negative charges and separate as atoms, and this accounts for the gold found in the ferric oxide attached to iron plates when these are used as anodes.

The deposition of gold from cyanide solutions has been carefully studied in connection with electroplating, and something bearing upon our subject may be learned from the experience gained in that process, although the purposes and conditions of the work in ore treatment are very different.

For example, G. E. Bonney states\* that with both gold and silver an excess of anode surface tends to enrich the solution, while an excess of cathode impoverishes it. This of course refers to the use of gold anodes, but nevertheless it points to the advantage of a large cathode area when the intention is to reduce the quantity of gold in the solution as far as possible.

Again, it has been found that to obtain a good adherent deposit in electroplating, the current must be roughly proportional to the quantity of gold per unit of solution. For a current of 10 ampères per square foot, the poorest solution used contains about 50 ozs. of gold per ton, and this is equal to 0.01 ampère per square foot for each dwt. in the solution.

Sherard Cowper-Coles obtained a bright coherent deposit from solution containing 20 ozs. per ton with three ampères per square foot = 0.0075 ampère per dwt. In ore treatment with, say, 4 dwts. per ton of solution, a current of 0.05 ampère per foot will give an adherent coating = 0.013 ampère per dwt.

As the relation of current strength to solution value is nearly the same through such a wide range, it may be fairly assumed that it will hold good

\* *Electroplaters' Handbook*, Whittaker & Co., 1898, p. 158.

with still poorer solutions ; and therefore to obtain a good firm coating of gold on the cathode, the current density should be less for extremely poor solutions, as for instance in slime treatment, than it is for the relatively richer ones dealt with in treating sands. But, as will be seen later, it is not always necessary or desirable to have an adherent deposit.

It has also been found in plating that with solutions poor in gold, the quantity deposited per unit of current is less than the theoretical amount, and, as might be expected, this loss of efficiency is enormously greater when dealing with the very poor liquors which are used in the cyanide treatment of ores and tailings. Thus, in the former case a current of 10 amperes per square foot will deposit about 40 grains per ampère-hour from a liquor containing about 100 ozs. per ton, while in the latter case from a 3 dwt. solution about 2 grains per ampère-hour may be expected, and in cases where the solution is practically exhausted the quantity may not exceed 0·2 grain. (For method of comparing current efficiency see page 115.)

#### THE ELECTRODES.

**Anodes.**—These may be broadly divided into three classes as follows:—

1. (a) Those soluble in the cyanide solutions ; (b) Those soluble in an electrolyte separated from the cyanide solution by a porous partition.

2. Those which are difficultly soluble in a cyanide solution.

3. Those which are practically insoluble.

1 (a). The advantage of soluble anodes over insoluble anodes is that a very much smaller E.M.F. is required to produce the same result. Thus, in one case with a soluble anode and a current of 0·5 volt, the amount of a solution decomposed was as great as with 4 volts when the anode was insoluble. This is due in the two cases to a difference in the contact resistance to the current, as it leaves the electrode and enters the solution, which is sometimes called ‘transfer-resistance.’ This is undoubtedly a physical effect, which must not be confounded with the phenomena of polarisation. With insoluble anodes, this resistance is said to be largely due to a condensed film of gas, which is only slightly removed by agitation of the liquid, but it is also accounted for by other actions of a more complex nature.

When the anode is in a solution of maximum solubility, transfer-resistance is generally at a minimum, and as the solution becomes more dilute, the transfer-resistance increases. With a zinc anode, which is soluble in a cyanide solution, the E.M.F. of the current need be very small compared with an iron anode, which is difficultly soluble, in order to decompose the solution. But in the cyanide process the solutions are so very weak and variable that it becomes difficult to regulate the E.M.F. to suit. If the E.M.F. is too great, the electrode becomes coated with a zinc compound, which does not dissolve as fast as formed, but accumulates, and may offer such a resistance as to stop the current altogether. On the other hand, if the E.M.F. be sufficient to overcome this resistance, an excessive quantity of zinc is corroded, and is re-precipitated along with the gold and silver at the cathode. Other soluble

anodes in highly dilute cyanide solutions are acted on in the same way, and produce insoluble coatings which are troublesome to contend with.

1 (*b*). In order to obviate some of the above difficulties, inventors have sought to use porous partitions between the solution containing the anode and that containing the cathode. Thus, N. S. Keith \* proposed to use porous pots containing anodes of zinc or iron in chloride or sulphate of ammonium. The strength of the anode solution could then be adjusted so as to keep the transfer-resistance much lower than if these anodes were in the weak cyanide solution. There is, however, a transfer-resistance between the solution in the porous pot and the cyanide, to be taken into account. The process is not impracticable, but it remains to be proved whether it can be economically applied or not.

2. Iron is the most important metal of the difficultly soluble class. In a strong cyanide solution with a weak current the compound formed is, for the most part,  $K_4FeCy_6$ , but in a dilute solution with a current of high E.M.F., such as is generally employed, the K and Cy ions are too few to carry the whole of the current, so that the water and other compounds act also as conductors. The result is that the iron for the most part is oxidised by the liberated oxygen forming hydrated oxides of iron, and to a smaller extent a number of compounds with the cyanides, of which Prussian blue is one. After being in use some time, iron anodes swell considerably in thickness, due to an accumulated crust of iron oxides.

3. Gas-carbon may be put into this class, but in all solutions in which oxygen is liberated at the anode, carbon disintegrates, forming a black substance called mellogen, with some carbonic oxide and carbonic acid.

Graphite also disintegrates, and becomes partly oxidised in the same way, but no mellogen is formed.

Peroxide of lead has been revived by E. Andreoli, who first prepared his plates by the method which Planté introduced in connection with electric accumulators. But his present method † is to place the lead plates in a solution of plumbate of soda, in which "they are rapidly coated with peroxide of lead, then withdrawn, washed, and placed in a strong KCy solution, where, under a heavy current, they become hard, and have a good crystalline appearance."

Bettel prepares lead anodes by painting their surfaces with peroxide of lead, or pulverised graphite incorporated with linseed oil.

If lead plates are properly peroxidised, they have good crystalline surfaces. In this state they are very slightly acted on by the cyanide solution, and last a long time. If, however, the peroxidising has not been properly done, the plates soon blister, even with a weak current (0.06 ampère per square foot), and fall to pieces.

Charles Butters was the first to successfully use the Andreoli peroxidised lead plates as the anodes on a commercial scale. The advantages over iron

\* *Jour. Inst. Elect. Eng.*, vol. xxiv. p. 242 *et seq.*, 1895.

† *Jour. Soc. Chem. Ind.*, xvi. 96, 1897.

were found very striking in the absence of sludge in the bottom of the boxes, and as there is no wasting away to provide for, these plates need not be very thick. Then, as the covering of sacking used with iron is unnecessary, and further, as there is no growth of adherent matter upon them, they can be placed closer to the cathodes. For actual results obtained with these Andreoli plates in practice see Table XXXIII., pages 150 and 151.

**Cathodes.**—Just as there is transfer-resistance to the current in passing from the anode into the solution, so there is transfer-resistance in passing from the solution to the cathode. Its magnitude varies with the nature of the electrode and the electrolyte, being generally greater than the resistance at the anode, although in some cases it is much less.

It was stated several years ago by A. von Gernet that the cathode should fulfil the conditions quoted below, and as this statement has been largely accepted and often repeated, it is worth while to discuss it.

"1. The precipitated gold must adhere to it.

"2. It must be capable of being rolled out into very thin sheets, to save unnecessary expense.

"3. It must be easy to recover the gold from it.

"4. It must not be more electro-positive than the anode, in order to prevent return currents being generated when the depositing current is stopped. The most suitable metal was found to be lead, which in the form of lead foil meets all requirements, and is therefore used in the Siemens-Halske process."\*

Of these so-called necessary conditions it may be said—

1. It really does not matter whether the gold adheres or not, provided that it can be readily recovered; with an insoluble anode the gold can be precipitated quite as conveniently and allowed to fall to the bottom of the box, while with the iron anode there is always so much gold in the sludge that it has to be treated, and therefore if the whole of the gold were in the sludge instead of partly on the cathode, it would mean treating the sludge *only*, instead of bullion *and* sludge. Therefore, technically, there is no advantage in an adherent coating, though financially with the Siemens process there is an advantage, in that the gold can be more quickly realised in most cases from the lead than from the sludge.

2. As this is merely a matter of cost, a cheaper material in somewhat thicker sheets would be equally good.

3. This is merely a corollary of No. 1, and disappears with it.

4. This is of little importance as regards the reason given, because it is quite simple and usual to break the circuit when the dynamo is stopped, thus preventing any return current. But as a matter of fact, lead is more electro-positive than iron in KCy solutions, and therefore does not fulfil this fourth requirement. See Von Oettingen's table on page 111. Therefore it is clear that these conditions are not of general application.

While offering the above criticisms upon the theoretical reasons given for the use of lead foil, we fully appreciate its practical value as a cathode and the

\* *Proc. Chem. and Met. Soc. of S. Africa*, vol. i. p. 30.

many other advantages of the Siemens-Halske process, which is dealt with in detail below (see page 142).

The following other substances have been proposed as substitutes for lead for this purpose.

Carbon in thin blocks would no doubt be quite effective electrically, but would be fragile to handle, and costly.

In 1894 Julian made application for a patent in the Transvaal for the use of cathodes made on the principle of those used in galvano-plastic work. The cathode consisted of metallic or other surface coated with a conducting substance. Thus iron coated with plumbago or waxed conducting substances were tried, the great advantage being that the bullion could be stripped off the cathodes and melted into bars without being contaminated with base metals. With rich solutions the process worked well, but when poor in bullion it became difficult to get a film deposited evenly, so that the effective electrode surface was diminished.

Another cathode on the same principle was employed, which consisted of cotton or other fibrous threads or woven material, rendered conducting by one of the well known methods. These passed over rollers up and down a number of times through each compartment of the precipitating box, using lead peroxide anodes. These threads were continuous in long lengths, and to remove the deposited metal they were drawn between rollers at the head of the box and cut off. A great advantage of this method is that a clean-up could be conducted very often, with only a short stoppage; and further, that the bullion obtained is not necessarily contaminated with other metals, as the bullion-covered threads have only to be smelted with borax and an oxidising agent. Of the woven materials a cheap open cheese cloth was found to be the best.

Zinc plates were tried by Andreoli, who was not satisfied with them, but preferred ordinary sheet iron or steel, from which he removed the gold deposit by dipping it into a bath of molten lead covered with oil. We have also used iron with success, and find that in order to get the best results all the oxide should be removed off its surface. This we have done by immersing the plates in the following mixture:—100 parts of water, 10 sulphuric, in which 1.5 of zinc is dissolved, and then adding 10 of nitric acid. If then washed in abundance of water and kept in an alkaline solution they remain nearly as bright as silver. In this state they receive a perfectly uniform deposit. Lead cannot be used to remove the deposit as this injures the iron surface for re-use, but if placed in narrow iron boxes containing 2 to 5 per cent. KCy solution, and connected with Daniell cells in parallel to form the anode while the box is the cathode, the gold and silver rapidly dissolve off without re-precipitating, leaving the plate perfectly bright and clean, and ready to go back into the precipitating box. The cyanide solution, when rich enough in bullion, is then evaporated in an iron vacuum pan, and the residue is simply melted in a plumbago crucible, the product being pure bullion and a slag of cyanide and cyanate. This slag being soluble in water, no shot metal need be left behind.

Aluminium was proposed by Sherard Cowper-Coles,\* who says that the gold can be deposited as a metallic sheet, and afterwards detached by stripping or rubbing. We experimented with aluminium as a cathode at the Rand Central Ore Reduction Company in 1893, and found that, owing to the difficulty of removing and collecting thin deposits, and for other serious reasons, they were unsuitable.

Amalgamated copper plates as cathodes have been used in various ways, as, for instance, in the Pelatan-Clerici process, described on page 157. They have also been in use by Julian, as described pages 3 and 4, by Keith and Andreoli, but the most scientific study in connection with them has been made by Dr T. K. Rose.†

The result of his experiments with solutions containing 10 ozs. of gold per ton is, that with a current of greater density than 0.03 ampère per square foot part of the precipitated gold is amalgamated, but the remainder is thrown down on to the mercury in the form of black powder, which will not immediately amalgamate unless sodium amalgam is added. This black powder can be removed by means of a feather. As the current density is decreased the proportion of black powder grows less, until at 0.027 ampère practically the whole of the precipitated gold is amalgamated, and the cathode remains clean. Dr Rose has apparently not investigated this powder to ascertain whether it is metallic gold or whether it is a compound of gold and cyanogen. These experiments certainly show that with 10 oz. solutions the gold is thrown down faster than it can amalgamate. But they do not lead to any conclusion as to what would happen with solutions containing a few dwts. or less per ton. It seems likely in the latter case, that the same or even a greater current density could be used without forming the black powder, because deposition takes place so very much more slowly from liquors which carry little gold. From a scientific point of view it would be interesting to clear up this doubt, for unless a high current density can be employed, there is little likelihood of amalgamated copper being used as a cathode for the complete exhaustion of low grade solutions, due to the following reasons:—

1. With a surface equal to that employed with lead, the cost of copper and mercury would be excessive.
2. The plates permanently retain a portion of the precipitated gold.
3. The plates become corroded and in time get destroyed, possibly by the joint action of the mercury and potassium, or sodium, electrically deposited from the solution.

In spite of these objections, however, such cathodes are in use in South Kalgurli mine in West Australia for the partial precipitation of the gold from slimes pulp, which is kept agitated in the precipitation tank, the pulp being afterwards filter-pressed.

\* *Trans. Inst. Min. and Met.*, vol. vi. p. 219.

† *Trans. Inst. Min. and Met.*, vol. viii. p. 369, 1899-1900.



## CHAPTER XXII.

### THE SIEMENS-HALSKE PROCESS.

IN the practical application of this process, so ably worked out by A. von Gernet and Chas. Butters, the anodes consist of sheet iron from  $\frac{1}{8}$  to  $\frac{1}{4}$  inch in thickness. These are sewn up in covers of medium quality hessian or other suitable textile material of fairly open texture, in order to prevent short-circuiting, which otherwise is very likely to occur as the plates increase in thickness, by the accumulation of iron oxide on their surface.

In the first installation at the Worcester mine in Johannesburg these anodes were 7 ft. long by 3 ft. wide. Afterwards, in order to decrease the number of electrical connections, larger plates were tried, up to a maximum of 8 ft. by 4 ft. 6 ins. But, as might have been anticipated, these large plates proved too unwieldy for convenient handling, and the present practice is to use small plates in all cases, and to vary the number according to the size of the precipitation box, as it is practically easier to deal with a large number of plates light enough for one man to lift than to have fewer plates of great weight. The increased number of connections is a matter of small importance, because each one has less current to carry. Also with small plates the current density is more uniform than with large ones.

A convenient size is from 4 to 6 square feet, so that each plate will weigh from 30 to 60 lbs. ; and in choosing the unit size, reference should be made to current market dimensions, so as to avoid waste in cutting. The standard size adopted by the Rand Central Ore Reduction Coy. in Johannesburg is 2 ft. by 1 ft. 8 $\frac{1}{2}$  ins. by  $\frac{3}{8}$  inch. In Australia we used plates 2 ft. 4 ins. by 2 ft. by  $\frac{1}{4}$  inch, and found them very convenient. At the top corner of each anode is brazed a strip of iron 1 inch wide by  $\frac{1}{8}$  inch thick to connect with the conductors from the dynamo. The strip and its junction with the plate are covered to prevent corrosion.

The cathodes consist of lead foil .0032 inch thick, weighing about 0.19 lb. per square foot. Sheets of this foil are hung from horizontal iron wires, by turning the top edge of each sheet over the iron wire, and sewing with fine copper wire. The supporting wires were at first fixed in a wooden frame, but this is now dispensed with. These wires are connected to the main conductor from the dynamo.

The earlier boxes were divided into compartments in the following way : The width of the box was made equal to the length of the iron plates, which were placed crosswise in the box. Water-tight joints were made between

TABLE XXXI.—Dimensions of Boxes and results obtained with the Siemens-Halske method of Electrical Precipitation.

Gold per ton after precipitation in grains.	...	...	trace	18	24	21	3
Gold per ton before precipitation in grains.	...	...	48	120	24	93	17
Specific rate of flow, i.e. cubic feet per minute per square foot of section.	...	0.041	...	0.059	0.039	0.066	...
Rate of flow in cubic feet per minute.	...	1.56	...	6.67	7.11	3.42	...
Length of column in feet.	...	23	...	24	44	48	48
Section of liquid column in square feet.	...	38	...	114	80	40	186
Cathode surface in square feet.	12672	5274	7680	...	...	...	...
Anode surface in square feet.	14784	5040	3240	13824	16896	9216	33024
Total cubic feet occupied by electrodes.	1512	770	1080	2736	3520	1920	8930
Cubic feet in boxes per ton of solution.	16.8	16	25	14.25	18.13	20.13	...
Total tons of solution in 24 hours.	120	70	50	300	320	154	...
Total cubic feet in boxes.	2016	1125	1249	4275	5800	3100	13950
Depth of each box in feet and inches.	4' 0"	4' 0"	9' 0"	5' 0"	5' 0"	5' 0"	5' 0"
Width of each box in feet and inches.	7' 0"	5' 2½"	4' 7½"	4' 9"	10' 0"	5' 0"	12' 0"
Length of each box in feet and inches.	18' 0"	18' 6"	30' 0"	30' 0"	29' 0"	31' 0"	30' 0"
Ampère per square foot.	0.047	0.06	...	0.04	0.04	0.04	0.04
Number of boxes.	4	4	1	6	4	4	3
Per cent. KCy in Solutions.	.15 to .05	.2 to .02	.15 to .05	.08 to .02	.08	.10 to .03	.017
Name of Works.	Worcester, .	Gibraltar, .	Crossus, .	May Consolidated, sand,	" slime,	Lancaster West, .	Ferreira, slime, .
							" "

some of the plates and the sides and bottom of the box, while sufficient space was left under the other plates to provide for the circulation of the solution. This method was found to be troublesome when cleaning up the sludge or attending to the anodes. In the later boxes wooden partitions were provided with alternate spaces for the downflow of the solution between the precipitating compartments. In these boxes the electrodes are placed parallel to the sides of the box, and the length of each compartment is made to suit the size of the anode plates. Further details will be given of these boxes when dealing with the construction of the apparatus used. Table XXXI. gives a number of particulars as to dimensions, etc. taken from actual practice, and includes some of the most recent installations.

It will be seen in the above table that in the earlier boxes used at the Worcester mine and at the Gibraltar mine in Australia, which were provided with alternate sheets of iron and lead of practically equal size, that about 16 cubic feet capacity was required for each ton of solution treated in twenty-four hours. This is considerably in excess of the size used for zinc boxes, and it will be interesting to notice the means by which the size has been somewhat reduced already, and to examine the possibilities of further reduction. At least three ways suggest themselves of effecting improvement in this direction, viz., first by increasing the *density* of the current; second, by putting the electrodes closer together, thereby enlarging the surface of both electrodes, and consequently increasing the *total* current; third, by increasing the *cathode* surface without increasing that of the anodes. (1) By merely strengthening the current the total amount of gold precipitated is very slightly increased; a larger proportion of it is, however, thrown down in the sludge, but the principal result is a greater decomposition of water, thereby liberating more oxygen at the anode, and so causing a larger waste of iron. With this process, therefore, it is of no advantage to increase the current with the object of diminishing the size of the boxes. The greatest current density found suitable in practice is 0.06 ampère per square foot of anode.

(2) By placing the electrodes closer together and using the same current density, the total current will be increased, and the gold precipitated in a given time, though not in proportion to the additional electrode surface which can be placed in the box. Now, the thickness of an anode covered with sacking is about  $\frac{3}{8}$  inch, so that if the lead foils actually touched the sacking there would still be nearly half an inch between the centres of the anodes. But in working, as the ferric oxide forms on the anodes it swells out to say  $\frac{5}{8}$  inch, and as some space must be allowed, for the passage of the liquid between the electrodes, it is evident that the anodes cannot be placed closer together than  $1\frac{1}{2}$  inch centres as the extreme practical limit. In the boxes as originally made, the anodes were at 3 inches from centre to centre, so that it seems possible by this means that the cubic contents of the boxes can be reduced to about one-half the original size. This only applies to small anodes, where there is no chance of the plates buckling, because with large plates the 3-inch spacing is necessary on account of the twisting and bending of the iron anodes.

(3) But the third method has hitherto received more attention, namely, the enlargement of the cathode surface only. This is done by cutting the lead foil into strips about an inch wide, and slightly crumpling the strips to prevent them sticking together, the result being a sort of long fringe of lead. Several sheets so cut are then hung on an iron wire frame about 3 inches wide and placed between each pair of anodes, which in this case are spaced at  $4\frac{1}{2}$ -inch centres. At present two sheets are usually hung together on each frame, so that the cathode surface is thereby doubled, while the anode surface is reduced in the ratio of 3 to 2. Under these circumstances, with the same current as before, the precipitation is increased, because, as stated on p. 136, the excess of cathode tends to impoverish the solution. In other words, the amount of gold precipitated depends upon the area of cathode surface as well as upon the current, and is independent of the anode surface. The reason for this has never been fully explained. But the increased efficiency of the box is not proportional to the enlarged cathode area; and although this method has been fully tried, with various numbers of lead sheets on each frame, the best result as yet obtained is that by doubling the cathode surface per cubic foot, the box can be reduced to about two-thirds of the size of the original type. As some parts of the lead are sensibly farther from the anode than other parts, the deposition of gold is less uniform; and by having so much more lead, either the resulting bullion must be poorer or the gold must remain longer in the boxes, thereby incurring a greater loss of interest. Some boxes are now being tried without enclosing the anodes in sacking. In order to avoid any possibility of short-circuiting, the anodes are at present placed at more than the usual distance apart. If wider spacing proves in practice to be really necessary, there does not seem to be any advantage in such an innovation. This brief review of the various methods tried is sufficient to show that the best possible arrangement of electrodes for this process has not yet been finally determined.

Table XXXII. shows the number of square feet of anode or cathode surface required for a given precipitation when working with electrodes of equal area. All the available figures have been plotted on squared paper, curves drawn through the points so obtained, and the table compiled by scaling the figures from the diagram.

Half this surface multiplied by the distance in feet between one pair of anodes will give the cubic contents occupied by the electrodes. By adding 50 per cent. to this for unoccupied space, the capacity required in the boxes is determined for any degree of precipitation for one ton per day, and this multiplied by the tons of solution to be treated in twenty-four hours will give the total capacity of the boxes in cubic feet.

As an illustration of the use of this table, we may take the following example:—

Let us assume that it is required to precipitate 200 tons per day of solution carrying  $3\frac{1}{2}$  dwts. (84 grs.) per ton down to 8 grains, which will represent the maximum demand in ordinary sand treatment.

From the table we get 90 square feet, and taking the anodes at 0.25 ft. centres, we have  $\frac{90}{2} \times 0.25 = 11.25$  cubic feet occupied by electrodes for each ton of solution. Adding 50 per cent. gives 16.875 cubic feet. Multiplying by 200 we get 3375 cubic feet as the total capacity required in the boxes for this particular case.

TABLE XXXII.—*Showing area of either Electrode required in the Electrical Precipitation of 1 ton of solution in 24 hours, when working with anodes and cathodes of equal surface.*

Original Value of Solution. Grains per ton.	Value of Solution after Precipitation, in grains per ton.							
	20	16	12	8	4	2	$\frac{1}{2}$	Trace.
84	48	58	70	90	112	133	154	176
80	47	57.5	69	89	111	132	153	174
78	46.5	57.3	68.5	88.5	110.5	131.5	152.5	173.5
76	46	57	68	88	110	131	152	173
72	45	56	67	87	109	130	151	172
68	44	55	66	86	108	129	150	171
66	43.5	54	65	85.5	107.5	128.5	149.5	170
64	43	53	64	85	107	128	149	169
60	42	52	63	84	105	126	147.5	168
56	40	51	62	83	103	124	146	167
54	39.5	50.5	61	82	102.5	123.5	145	166
52	39	50	60	81	102	123	144	165
48	37	48	58.5	79	100	121	142	163
44	34	45	57	77	97	118	139	160
42	33	44	55	76	95.5	117	137.5	158.5
40	32	42	53	74.5	94	116	136	157
36	29	39	50	71	91	112	132	153
32	25	35.5	46	67	87	108	128	149
28	21	31.5	42	62	82	103	123	144
24	15	25	35	56	76	96	116	137
20	...	16	27	43	66	87	107	128
16	...	...	...	34	54	76	96	117
12	...	...	...	...	40	61	82	103
8	...	...	...	...	22	43	64	86
4	...	...	...	...	...	...	43	66
2	...	...	...	...	...	...	...	43

*Note.*—Find horizontal line corresponding to known original value and vertical column under the residual value. At the intersection of these two is the area required.

When the cathodes are each formed of two sheets of lead in strips the table can still be used, but the actual capacity required in that case is, as already stated, about two-thirds of that calculated in the above manner.

From the table it may be seen that the area required to completely precipitate a solution carrying 72 grains per ton is very little greater than is necessary for a solution worth 36 grains, and this illustrates the fact, that the rate of precipitation diminishes very rapidly as the liquor becomes poorer. This is still more clearly shown in fig. 26, in which we have plotted the percentages of

gold precipitated at equal intervals of time, as calculated from results of precipitation tests made by J. R. Williams, and published by the Rand Central Ore Reduction Coy.

**Circuit and Connections.**—The main current is generally carried along the sides of the box by copper strips, provided with branch strips, which are carried along the partitions of the several compartments. The iron strips of the anodes and the supporting wires of the cathodes are sometimes connected to these copper strips by means of small bolts and nuts, but in other cases the branch conductors are replaced by mercury in the following manner. A piece of hard wood is fixed on the partitions, and provided with a longitudinal groove  $\frac{3}{4}$  inch deep by  $\frac{1}{4}$  inch wide. This groove is nearly filled with mercury, into which the wires from the electrodes dip. This method is more

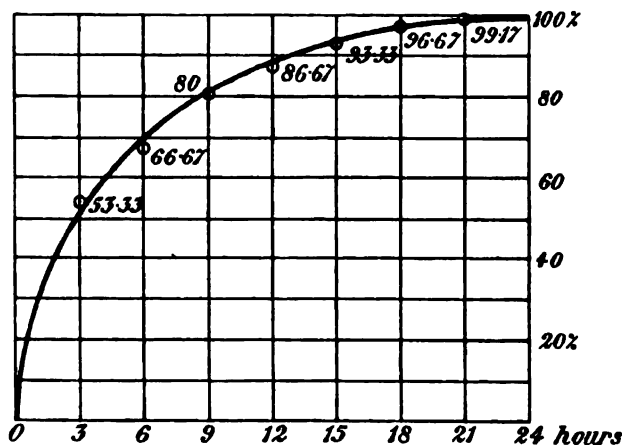


FIG. 26.—Rate of Precipitation of Gold.

frequently used for the cathodes than for the anodes. For the same purpose an iron tube filled with mercury has been tried by Carlin. A. von Dessauer suggests a grooved iron bar, or a bar with holes at intervals filled with mercury. He gives\* the following measurements by Dr Pauli, to show how the loss of voltage and consequent variation of current density is lessened by using metal instead of wood.

(a) Wooden trough with mercury. Terminal E.M.F. 3.82 volts, maximum decrease in 8 feet 0.71 volt. This means that if the current density on the first plate were 0.05 ampère per sq. ft., it would be only 0.041 on the last electrode attached to the same trough.

(b) With  $\frac{1}{2}$  inch brass tube 8 ft. long filled with mercury the figures were: Loss of E.M.F., 0.015 volt; loss of energy, 27 watts as against 1300 watts = 1.74 h.p. with the wooden trough.

The original method of arranging the circuit was to use four boxes in series,

\* The Carlin tube—*Jour. Chem. and Met. Soc. S. Africa*, vol. ii. No. 4, p. 63.

and to place all the plates of each box in parallel, as shown diagrammatically in fig. 27.

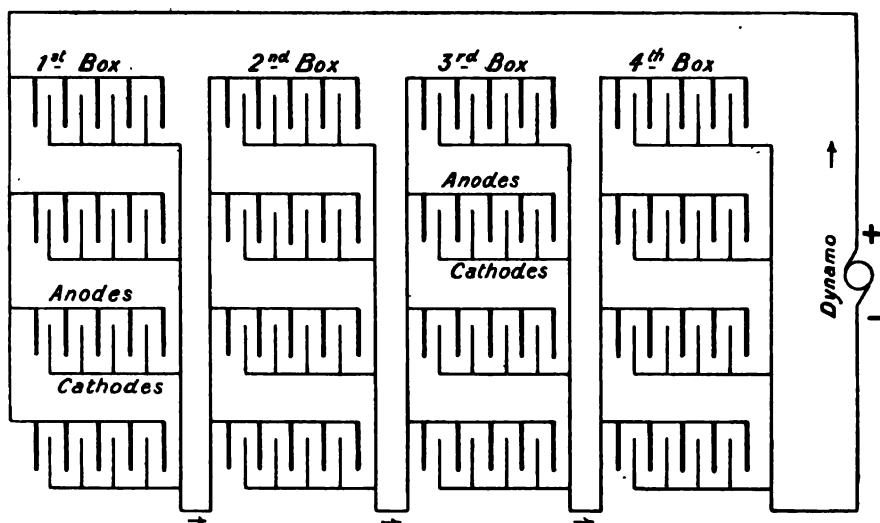


FIG. 27.—Arrangement of Circuit with Electrodes in Parallel and Boxes in Series.

In some of the later plants the boxes were placed in parallel, but the plates in each box were in series. Thus the anodes of the first section were connected to the positive pole of the dynamo, while the cathodes were connected to the

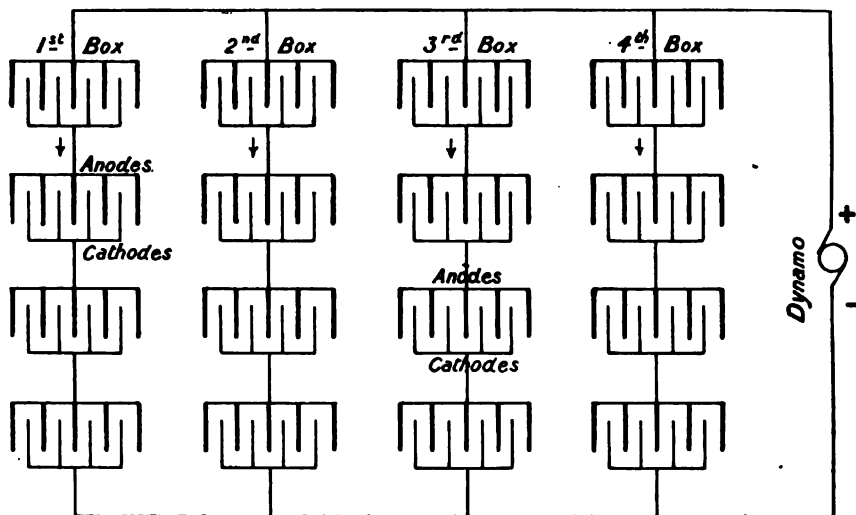


FIG. 28.—Arrangement of Circuit with Electrodes in Series and Boxes in Parallel.

anodes of the following section, the last set of cathodes in each box being, of course, connected to the negative pole of the dynamo, as shown in fig. 28. We

have always used the first and simpler method with satisfactory results, and there seems to be no particular advantage in the second arrangement. When the boxes are in parallel, a separate current indicator should be provided for each box. When in series, fig. 29 shows a simple method of connecting the volt meter through a five-way switch, so that the voltage can be read on any number of boxes, or on the line wire alone, or on the dynamo. An alarm bell or visible indicator is necessary to give notice if the current by any means is reversed in direction. Messrs Siemens and Halske of Berlin make a shunt current regulator, which also reverses the polarity of the dynamo when required.

**Iron Consumption and Sludge Production.**—The iron consumed per ton of solution treated will depend upon the total anode area and the current, which again depend upon the original value of the solution and the degree of impoverishment required. In most cases the amount of iron used will be between 0.33 and 0.67 lb. per ton of solution, or say an average of 0.5 lb. Assuming that half of this loss is in worn-out plates and that the other half is converted into oxide, the amount of sludge to be expected is 0.35 lb. per ton. In practice, however, the amount of sludge collected does not usually exceed

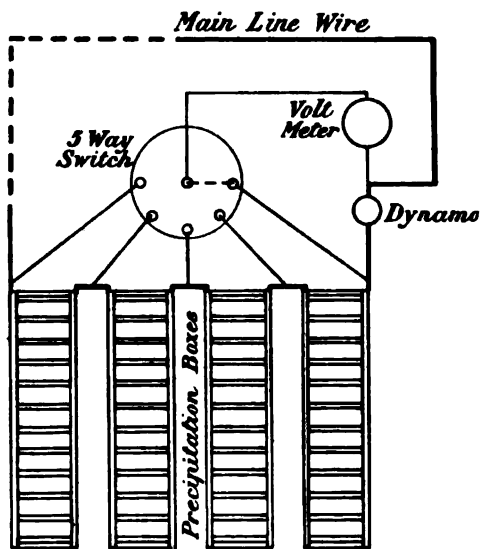


FIG. 29.—Arrangement of Switch.

0.2 lb. per ton of solution. Some of the iron dissolves, becoming  $K_4FeCy_6$ , and some of the ferric oxide formed is carried off in suspension in the water, and is deposited in the solution tanks and ore vats. A little gold will be carried with it, but this is probably re-dissolved in the subsequent operations.

**Lead Consumption.**—The amount of lead foil used depends almost entirely upon the relative value of the liquor treated, as compared with the grade of bullion produced. Thus, if the average extraction per ton of solution



TABLE XXXIII.

Nov.	No. of Boxes working.		Tons Solution 24 hours.		Solution Values Box 3 and 6.				Solution Values Box 1, 2, 4, 5.			
			1, 2, 4, 5.		Entering.		Leaving.		Entering.		Leaving.	
	Weak.	Strong.	Weak.	Strong.	Au \$.	Ag. oz.	Au \$.	Ag. oz.	Au \$.	Ag. oz.	Au \$.	Ag. oz.
1	4	2	372	288	2.76	3.32	.18	.20	1.32	1.66	.12	.16
2	4	2	384	288	2.76	3.18	.16	.19	1.24	1.61	.15	.15
3	4	2	312	216	2.80	3.60	.16	.18	1.04	1.35	.12	.14
4	4	2	384	216	2.64	3.10	.18	.19	1.12	1.50	.12	.14
5	4	1 $\frac{3}{4}$ *	384	216	3.40	4.11	.18	.26	1.04	1.45	.08	.14
6	4	2	384	216	3.28	4.28	...	...	1.12	1.49	...	...
7	4	1 $\frac{3}{4}$	384	216	3.48	4.72	.34	.43	1.08	1.53	.15	.19
8	4	2	384	216	3.32	4.09	.34	.38	1.12	1.45	...	...
9	4	2	384	216	3.24	4.43	.46	.57	1.08	1.45	.15	.15
10	4	2	384	216	3.04	3.94	.34	.45	1.20	1.59	.14	.19
11	4	2	384	216	2.80	3.84	.20	.27	1.28	1.70	.16	.19
12	4	2	384	288	2.36	3.04	.16	.23	1.32	1.65	.14	.14
13	3 $\frac{1}{2}$	2	384	288	2.96	3.64	.14	.19	1.24	1.65	.13	.18
14	4	2	384	288	...	...	...	...	...	...	...	...
15	4	2	384	288	...	...	...	...	...	...	.15	.18
16	3 $\frac{1}{2}$	1 $\frac{3}{4}$	320	240	3.20	3.19	.28	.23	1.16	1.44	.13	.16
17	4	2	384	288	3.68	4.78	.28	.38	1.04	1.30	.16	.16
18	4	2	384	270	3.80	4.58	.30	.36	1.04	1.36	.12	.19
19	4	2	384	288	2.96	4.35	.14	.22	1.00	1.29	.15	.16
20	4	2	384	288	2.52	3.34	.20	.25	1.08	1.47	.17	.17
21	4	2	368	288	2.20	2.41	.20	.26	1.04	1.38	.06	.10
22	4	1 $\frac{3}{4}$	312	216	2.52	2.45	.20	.21	1.04	1.41	.08	.09
23	4	2	273	216	2.60	2.77	.18	.18	1.00	1.35	...	...
24	4	2	312	288	2.68	2.55	.24	.22	1.00	1.38	...	...
25	3 $\frac{3}{4}$	2	312	288	2.40	1.78	.22	.23	.96	1.30	.08	.09
26	4	2	384	288	3.28	3.59	.24	.26	1.08	1.39	.12	.12
27	4	2	384	240	3.20	3.97	.34	.37	1.04	1.34	.16	.19
28	4	2	384	240	2.84	4.10	.32	.36	1.12	1.42	.14	.15
29	4	2	408	240	...	...	...	...	...	...	...	...
30	4	2	408	240	2.92	3.52	.30	.30	.96	1.21	.14	.13
Average					2.95	3.58	.24	.28	1.10	1.45	.13	.15

\* The fraction indicates that a portion of the box had been thrown out of the circuit on that day.

TABLE XXXIII.—*continued.*

Current Box 1, 4.		Current Box 2, 5.		Current Box 3, 6.		Ampères per sq. ft. Anode Surface.			Anode Surface square feet.				
Volts.	Amps.	Volts.	Amps.	Volts.	Amps.	3, 6.	2, 5.	1, 4.	3, 6.	2, 5.	1, 4.		
35	243	27	241	26	488	•56	•28	•35	6950	6800	6800		
34	243	27	243	27	490	•56	•28	•35					
34	241	27	242	27	494	•56	•28	•35					
35	242	27	242	27	497	•56	•28	•35	Output for Month.				
32	199	26	240	20	430	•49	•28	•29					
33	200	27	240	25	484	•56	•28	•29					
32	200	26	238	21	449	•51	•28	•29					
32	203	26	242	25	486	•56	•28	•29					
32	207	26	241	25	485	•56	•28	•30				Box.	Oz. Au.
32	205	26	242	25	484	•56	•28	•30	6	205	5134	2300	
30	187	26	244	25	484	•56	•28	•27					
30	181	26	247	25	482	•56	•29	•26					
30	183	23	245	26	488	•56	•29	•26	3	771	20000	500	
30	184	27	243	26	484	•56	•28	•27					
32	197	26	245	26	487	•56	•29	•29					
32	194	26	242	26	485	•56	•28	•29	2	206	5396	6700	
32	200	26	245	25	492	•56	•29	•29					
32	200	26	245	25	487	•56	•29	•29					
32	201	26	242	25	489	•56	•28	•29	4	43	1054	1530	
32	200	26	243	26	480	•56	•28	•29					
32	197	26	243	26	480	•56	•28	•29					
32	195	26	245	26	480	•56	•29	•29	5	85	956	1110	
32	200	26	240	26	465	•53	•28	•29					
32	200	26	240	26	450	•51	•28	•29					
32	195	26	245	25	450	•51	•29	•29	Total,	1439	37382	18840	
32	200	26	235	25	450	•51	•28	•29					
32	199	26	243	25	448	•51	•28	•29					
32	200	26	238	25	447	•51	•28	•29					
32	200	26	240	25	450	•51	•28	•29					
32	200	26	245	25	445	•51	•24	•29					

is 2 dwts. and the bullion contains 3 per cent. of gold, the quantity of lead consumed is 0.22 lb. per ton. There is also waste of lead in cutting and variations in grade of bullion to be allowed for, and we have found in practice that under the above conditions the lead used is from 0.25 to 0.35 lb. per ton of solution. This will serve as a guide in estimating the consumption under any other specified conditions.

**Gold in Bullion and Sludge.**—The following results from practical work will show how the values recovered are divided between the bullion, sludge, and slags.

*Example a.*—With clean quartz ore.

Gold in bullion, . . .	2603.63 ozs.	93.36 per cent.
„ „ sludge, . . .	185.11	6.64
	<hr/> 2788.74	<hr/> 100

*Example b.*—Coppery ore.

Gold in bullion, . . .	1323.64 ozs.	86.42 per cent.
„ „ sludge, . . .	185.94	12.15
„ „ slags, . . .	21.91	1.43
	<hr/> 1530.49	<hr/> 100

In the first case the gold called for by assay was 2832.16 ozs., so that the actual recovery was 98.47 per cent. of the theoretical. In the second case the gold as per assays should have been 1488.53 ozs., so the actual recovery was greater than the theoretical, due no doubt principally to inaccurate tonnage measurements.

**Modifications in Electrical Precipitation.**—Chas. Butters made, as already stated, an important change in electrical precipitation by introducing, instead of iron, lead peroxide anodes, prepared by Andreoli's method, which did away entirely with the sludge that collected in the bottom of the box. He further made an important change by using a high density current instead of the low density necessary with the Siemens-Halske process.

As pointed out, when using iron anodes the greatest current density recommended is 0.06 ampère per foot square of anode, in order to avoid an undue consumption of iron. When, however, anodes of lead peroxide are used, the density of the current may be many times that amount without causing loss of anode.

**Physical Effect of a High Density Current.**—It has been already stated at page 123 that a low density current deposits a hard firm film at the cathode, whereas a high density current deposits a loose slimy or powdery precipitate. Butters conceived the idea that by precipitating the bullion as a loose slime it might be then wiped off, and thus obviate the necessity of destroying the cathode, as is done in the Siemens-Halske process. In order to make this possible an anode other than iron had to be employed. Lead peroxide was found to meet all the requirements necessary.

Butters uses a current density of about 0.28 to 0.56, or nearly ten times as great as practicable with iron anodes. The bullion separates as a

dark loose slime, and this is removed by simply running a wiper over the surface of the electrode while in position, at intervals, as the deposit accumulates. This metallic slime then falls to the bottom of the box, where it remains until the clean-up. Some of the advantages of the high density current are, that there is less irregularity in the results, and that the plant requires less supervision than the older method with a low density current. The bullion being metallic slime is in excellent condition for the refiner. The chief disadvantages are that the cost of power is increased and the quantity of cyanide destroyed is greater.

Table XXXIII.\* is a complete record of an ordinary month's work in the precipitating boxes at the Minas Prietas Works in Mexico, where Butters' modifications have been introduced, and have been working successfully for over two years.

\* We are indebted to Mr Charles Butters for this table, who has also kindly placed at our disposal a large number of other tables and practical results, being the work of members of his staff.

## CHAPTER XXIII.

### OTHER METHODS OF PRECIPITATION.

**Precipitation by Charcoal.**—According to the laws of electro-chemical action already discussed, carbon *per se* cannot precipitate gold, because it is strongly electro-negative to the latter in a KCy solution. That charcoal is, however, a practical precipitant for gold from cyanide solutions has been demonstrated in several plants in Victoria, chiefly under the supervision of W. B. Gray, of Maldon, but no satisfactory explanation of its action can be given. Although it is usual to refer the effect to the presence of hydrocarbon gases in the charcoal, or to the decomposition of the double cyanide of gold and potassium in some unexplained manner, neither of these suggestions throws any real light upon the subject.

The following quantitative results by S. B. Christy \* are of special interest because so little scientific work has been done in this matter. Two solutions were prepared, each containing 20 mgs. of gold in 2000 c.c. of water, equal to 5 dwts. 20 grs. per ton. One (a) contained no free KCy, the other (b) contained 0.2 per cent.

Two glass tubes 2 cms. diam. by 10 cms. long were used for precipitation, each containing 10 grammes of dry charcoal. The first 800 c.c. of each solution required nearly two days to filter, after which (a) contained no gold and (b) only a trace. The next 1200 c.c. took nearly five days to filter, and the total filtrates contained (a) 0.01 and (b) 1.40 mgs. respectively.

The charcoal filter (a) contained 20.03, while (b) contained 18.70 mgs., the actual extractions from these figures being 99.95 and 93.03 per cent. respectively.

An important point brought out by these figures is that the efficiency of the charcoal had already been impaired, for the recovery from the first portions of the two solutions was 100 and 99 per cent. respectively, while the second portions only yielded 99.91 and 88.39 per cent. For the aqueous solution (a) the difference is so slight that it is inconclusive, but the difference in the case of the cyanide solution (b) is very marked.

Prof. Christy mentions that, in accordance with the above figures, the charcoal from solution (a) carried 0.2 per cent. of gold, and from (b) 0.187 per cent. He concludes that "these figures seem to represent the limit of enrichment at which the charcoal ceases to act in either case, the limit being lower in the presence of free cyanide." He also found, with richer solutions carrying

\* "Solution and Precipitation of Cyanide of Gold," *Proc. Am. Inst. Min. Eng.*, vol. xxvi. p. 746 *et seq.*

about 30 ozs. to the ton, that the charcoal from an acid solution contained 0·21 per cent. of gold, and from an alkaline one containing some free cyanide the charcoal carried only 0·091 per cent. of gold.

If the gold is deposited in a metallic state, as it is from chlorine water, it is easy to understand why there is a limit to the amount carried by the charcoal when free cyanide or any other solvent for gold is present. For as soon as any gold is precipitated, a gold-carbon couple is thereby formed, which, of course, tends to re-dissolve the gold. As the gold surface increases, the rate of dissolution may in time become equal to the rate of precipitation. This explanation, however, does not apply in cases where no solvent for gold is present. It has been stated, moreover, that by reheating the charcoal, its power of precipitation is to some extent restored. We find by calculation, from records of actual work, that the charcoal when cleaned-up carries gold to the extent of 0·107 to 0·48 per cent. of its own weight. Differences of this kind at different plants are not surprising, as it is probable that the limit to the amount of gold deposited, depends rather upon the available surface in the pores of the charcoal than upon the weight of it, and in practice only the lighter and more porous portions are used, these being separated from the denser pieces by washing.

Taking the mean of the last two figures, namely, 0·29 per cent., it follows that for every ounce of gold recovered, about 24 lbs. of charcoal must be handled. Or for a clean-up of, say, 1120 ozs., twelve tons must be burnt, leaving, say, one ton of ash to be smelted.

An interesting fact is mentioned by John I. Lowles,\* namely, that "the gold caught by the top filter of a set is considerably purer than that caught in the lower ones. Frequent tests show this to be quite characteristic, steady diminution in the value of the gold taking place downwards through a set," and in the bottom tub the bullion consists almost entirely of silver.

The same writer gives the following flux for smelting alkaline ash from the burnt charcoal.

10	parts ash
15	" borax
6·67	" tailings.

In the colony of Victoria, where this method has been most frequently used, the practical procedure is generally as follows: The charcoal is crushed through a small pair of toothed rolls; the fine stuff is separated by sieving and thrown away, the harder portions are eliminated by washing, and the remainder is put into small tubs about 2 ft. in diameter inside and slightly deeper. These are placed on the floor of the precipitation house in from three to six parallel rows, each row being a few inches lower than the preceding one. Each tub has an earthenware pipe about 5 in. in diam. placed vertically in the centre of it, also filled with charcoal. The solution to be precipitated is divided equally between the tubs in the top row, and from each of these it

\* "On Charcoal Precipitation from Auro-Cyanide Solutions," *Trans. Inst. Min. and Met.*, vol. vii. p. 192.

flows successively through the tubs below, downwards through the central pipe of each, and upwards through the outer charcoal in the tub. A lid is provided over this, to prevent the pieces of charcoal from floating on the liquor and passing away.

At suitable intervals the top tubs are removed for cleaning-up, and each row of tubs is then transferred to the next higher position, while a set containing fresh charcoal is placed in the lowest row. Hence the custom of using such small tubs. It is evident that a considerable amount of labour is required for this purpose, for in a plant treating eighty-four tons of solution daily, there are, according to John I. Lowles, in the paper above quoted, 198 tubs, of which at least forty-eight are moved every day, this requiring two men.

The charcoal is preferably burnt in a Turnbull furnace. This has a conical combustion chamber, having a cover at the top provided with a feed door and a small grate near the bottom, while below the fire bars is an ash-pit which can be locked up. The outlet of the flue from the combustion chamber is placed under water in a separate compartment which has an outlet above the water level, provided with a steam ejector or other apparatus to maintain a draught. Thus all the products of combustion are caused to pass into and through the water, with a view to preventing any loss of gold by volatilisation or dusting. A small fire is lighted in the grate at the beginning of the operation, but afterwards the charcoal continues to burn without assistance, fresh supplies being fed in by the door in the cover as required.

In order to facilitate comparisons of this method of precipitation with others, we have calculated the following figures from published data of various plants:—Cubic feet of charcoal required in tubs per ton of solution treated daily, 7 to 14. Specific rate of flow, from 0.016 to 0.024 cubic foot per minute for each square foot of sectional area.

Conversely, each ton of solution per day requires from 0.9 to 1.4 square feet of section in the column of charcoal.

Length of column, 6 to 12 feet, *i.e.* from three to six tubs in series.

**Precipitation without separating the Solution from the Ore.**—The first cyanide process patented in 1867 by J. H. Rae belongs to this class, and is described on page 1. In 1887 Wanliss and Julian devised a process by which the gold was dissolved out of the ore and precipitated without separating the solution, as described on page 3. In 1889 we worked on a commercial scale at the Wemmer Gold Mining Company, Witwatersrand, Julian's electro-chemical process, which belongs to this class, and is described on pages 3 and 4.

The Pelatan-Clerici process, as described by Louis Pelatan,\* is practically identical with the second stage of Julian's electro-chemical process, in that the ore and solution are agitated together by means of rotating iron anodes over mercury-covered copper-plate cathodes. Pelatan, however, uses from

\* *Rev. Un. des Mines*, Tome xi., 1897.

two to ten kilos of common salt per ton of ore, in order to increase the electrical conductivity of the pulp.

Several Pelatan-Clerici plants were put into operation in America, and the following particulars of practical work are taken from a description of a fifty-ton plant at the De Lamar mill, Idaho, by D. B. Huntly.\*

The plant consisted of five agitating vats 8 feet 6 inches diameter and 8 feet deep, and ten tanks to collect the ore as crushed, measuring 8 feet 8 inches diameter and 4 feet deep, with dynamos and a few other appliances. The stirrer in each agitating tank consisted of a vertical hanging shaft carrying four arms, provided with a number of vertical pegs similar to those often used in amalgamating settlers. To each arm is attached a sheet iron anode 4 feet long by 14 inches wide and  $\frac{1}{2}$  inch thick. The amalgamated copper cathode covered the whole of the bottom. The ratio of the area of the anode to cathode was therefore about 19 sq. feet to 57 sq. feet.

The ore was crushed through a 30-mesh screen, with 120 tons of water to 100 of ore, and the strength of the cyanide solution used was 0.125 to 0.15 per cent. The time of treatment was  $11\frac{1}{2}$  hours.

The theoretical extraction on 871 tons was given as follows:—

Average original value,	.	.	.	12 dwts.	4	grs.	gold.
„ residue,	.	.	.	2	„	8	„ „
„ extraction,	.	.	.	9	„	20	„ „

equal to 80.8 per cent., and the silver was 79.5 per cent., but the actual recovery of the bullion was 70.1 per cent. After the process ceased working the copper plates yielded \$9000 bullion, which nearly accounted for the difference between the theoretical extraction and actual recovery. The loss in mercury was 0.32 lb. per ton ore.

Gilmour and Young worked a process at the Santa Francisca Gold Mining Company in Nicaragua, which was described in a paper read by A. James before the Institute of Mining and Metallurgy. The following extract † describes the process.

“The slime is then charged into a 5 ft. Boss pan in 2-ton lots, with 100 gallons of water, so as to form a very thick pulp. Mercury is then added until the globules can be seen circulating in the pulp, and then the required amount of KCy. After two hours' running, about 10 lbs. of mixed zinc and copper amalgam are added, and the pan run four hours longer. This precipitates the gold very completely, only 10 grains ordinarily remaining in solution, and this has been reduced to 1 grain per ton of ore. The contents of the pan are then discharged into a 7 ft. settler and the mercury recovered in the usual manner. About one hour is sufficient for the solution of the gold in the slimes. Slimes assaying  $1\frac{1}{2}$  ozs. gold and  $4\frac{1}{2}$  ozs. silver show an extraction of over 90 per cent. The works have four pans and two settlers.” It is stated that for a capacity of 50 tons a day 24 h.p. are required.

\* *Eng. and Min. Jour.*, p. 155, 1897.

† Extract taken from *Min. Ind.*, vii. p. 334.



The time allowed to precipitate the bullion from so thick a pulp appears to us exceedingly short, as also does the time allowed for the dissolution of the gold and silver.

**Molloy's Process.**—This process was patented in the Transvaal in 1892 by B. C. Molloy, where it was employed for some months. It consists in passing the gold-bearing solution over a surface of mercury in which sodium was continuously deposited. An ingeniously devised apparatus was employed, which consisted of a large shallow tray covered with mercury, down the centre of which was a narrow compartment, with sides that dipped slightly into the mercury. In this compartment was placed a strong solution of sodium carbonate which was electrolysed, the mercury at the bottom being the cathode, and the anode was of peroxidised lead. As the sodium deposited and accumulated in the mercury within the compartment it diffused to the mercury on the outside, over which a constant stream of gold-bearing solution was kept flowing. The gold-bearing molecules dissolved as they came in contact with the sodium amalgam surface were decomposed and gold amalgam formed.

The process was ably developed by Dr A. Simon, who employed it on a working scale in the treatment of tailings on Witwatersrand in 1892–3.

The weak point in the process lies in the difficulty of getting a sufficiently large electrode surface, for the solution to come in contact with, without unduly increasing the amount of mercury handled. Thus, if the area of a surface required to reduce a 5 dwts. solution down to 1 dwt. was 1, that to reduce a 1 dwt. solution down to 5 grs. would have to be somewhere about 25 with the same rate of flow, showing how enormously the area has to be increased as the solution becomes impoverished. This difficulty may, however, be got over, to some extent, by agitating the solution rapidly over the sodium amalgam surface as it flows, by mechanical means. The efficiency of sodium amalgam as a precipitant is small, but when employed in a dilute state, as in the Molloy process, it compares favourably with the electrical precipitation processes of to-day. Sodium amalgam is said to act on KCy in solution forming complexes, but in dilute working solution this action is inappreciable.

**Moldenhauer's Process.**—A process for the precipitation of gold and silver from cyanide solutions was patented in 1893 by Carl Moldenhauer. In the same year Julian conducted some experiments on a large scale at the Rand Central Ore Reduction Company on behalf of the Deutsche Gold und Silber Schiede Anstalt, using this process for precipitating instead of zinc.

The aluminium was first used in the form of plates, placed vertically at a distance of  $\frac{3}{4}$  inch apart, in a precipitating box divided into compartments, similar to that used in electrical precipitation. The result was, however, unsatisfactory, as a 6 dwts. solution at the head of the box gave a 2 dwts. sump solution. Aluminium shavings were next tried, when the resulting sump solution was lowered to about 10 or 12 grains.

It is necessary for efficient precipitation that either free alkali or free acid should be present in the solution. For obvious reasons the latter is difficult to

apply, if not impracticable, in the cyanide process. It was found after the process had been at work some weeks that alumina began to deposit on the shaving, which retarded precipitation of the gold. This was easily rectified by the addition of a little alkali at the head of the box, but as time went on the ever increasing quantity of alumina in the working solutions became more and more troublesome. Attempts were made to remove the alumina by precipitating it in the solution tanks, and then leave it behind in the ore, but it was found that whenever alumina was precipitated in a cyanide solution it also carried down a large proportion of the free cyanide. Methods have been suggested for overcoming the difficulty, but at a prohibitive cost.

**Pflegger's Process.**—This process was patented by Johannes Pflegger in 1895, and was for a short time in use on a working scale in the Transvaal, but we have no particulars of the results. It consists in employing a short-circuited simple cell, having zinc anodes in alkali or cyanide and iron cathodes in the gold-bearing cyanide solution, the two solutions being separated by means of a porous membrane.

The apparatus consists of a long narrow trough-shaped box, divided into compartments like an electrical precipitation box, but also having longitudinal divisions made of a special cloth forming the porous membranes. The divisions through which the gold solution flows is only 1 inch wide, and contain the cathodes. On either side of each narrow division are wider ones about 3 inches, and contain rods or bars of zinc in a 1 to 5 per cent. caustic alkali or cyanide, or a mixture of these. Each cathode is built up of five sheets of iron wire gauze, of about 12-mesh, placed parallel to each other at a distance of  $\frac{1}{2}$  inch apart, so that it practically filled its compartment, forming an interstitial cathode of extensive surface. The zinc anode is simply short-circuited with the iron gauze cathode, and the gold solution is allowed to flow through its compartments at a fairly high velocity.

This process was developed by Julian and worked on a full size experimental scale for three months on behalf of the Deutsche Gold und Silber Schiede Anstalt and he obtained remarkably good and uniform results. The original solutions entering the box varied from 3 dwts. to 1 oz. per ton, and the residuary solution consistently assayed only a trace to a few grains. One noteworthy advantage of this process over others, is that by intercalating electrical measuring instruments in the circuits the operator can estimate fairly well what the results will be.

The porous membranes consist of a cloth made from hard spun ramie fibre, woven fairly closely. This material answers perfectly, and improves with age.

The electro-motive force of the cell is between 0.6 and 1 volt. Being small, it is necessary that as little resistance as possible should be introduced into the circuit. For this reason, when precipitating from strong salt solutions, the extractions are generally rather better than when only traces of salts are present, and it is desirable that the solution should be kept flowing at a constant rate. If the flow stops altogether, some gold is apt to re-dissolve. The strong cyanide or alkali in the anode divisions is caused to circulate slowly,

and the residuary solution may be added to the solution tanks to bring up the requisite strength as required.

A precipitating box 35 ft. long, 3 ft. 3 in. wide, and 3 ft. 3 in. deep, containing eight rows of cathode divisions, is capable of dealing with 8 tons of solution per hour. At this speed the percentage precipitation was found to be a maximum.

The gold is removed from the iron cathodes by dissolving it off in a small quantity of a strong cyanide. This is conveniently done by placing the electrodes in a small square iron tank, capable of holding about a dozen electrodes at a time, and the dissolution may be assisted by blowing air through the solution, or by heating it from outside with steam to near boiling. The resulting rich solution of  $\text{KAuCy}_2$  is evaporated to dryness in an iron vacuum pan, and the residue is simply heated in plumbago crucibles to the temperature of molten gold. The product formed, cyanide and cyanate, is dissolved in water, and the particles of bullion are collected together and barred. In most cases the bullion is about equal in fineness to the amalgam gold from the mill.

## CHAPTER XXIV.

### CLEANING-UP, REFINING, AND SMELTING.

BEFORE beginning to clean-up, the zinc box is sometimes charged for an hour or two with strong cyanide solution, the object being to loosen the adhering gold on the shavings as much as possible. The next procedure is to displace the cyanide solution with clean water. Sometimes a little acid and alum is added to the water to facilitate settlement of the gold slimes. The zinc shavings are then carefully lifted and shaken in the water in the compartment, and are next placed over the adjoining compartment, where they are sprayed with a jet of clean water. After draining, the washed shavings are removed to a special water-tight tray, while the next compartment is being proceeded with. All the fairly long pieces of zinc are taken out and placed together for re-use, but the short stuff is allowed to fall to the bottom of the compartment.

In small works, the water in each compartment is allowed to settle, and when fairly clear is drawn off by means of a syphon or hand pump to a settling tank.

The turbid water and slimes left in each compartment is then removed to a clean-up vat by lifting it out in pails or by washing it out through a plug-hole, when provided, in the bottom of each compartment, and carried away by a launder to the clean-up vat. A few pounds of alum are often added to the slime, which is then given twenty-four to forty-eight hours to settle, when the clear supernatant liquid is drawn off.

In large works it is now usual to pass all the water decanted or pumped at the different stages through a small filter press, thereby obviating the loss of time necessary for settlement. Also, when the coarse zinc is separated, the slimes are sometimes filter pressed. The separation of the coarse zinc is in some works effected by rubbing the slimes through sieves, 20- to 50-mesh, but in others the whole of the zinc is dissolved out with sulphuric acid before filter pressing.

The clean-up occurs at intervals of a week to a month, but usually once a fortnight is preferred.

**Treatment of the Zinc-box Slimes.**—We have now arrived at the end of the first stage of the clean-up, whereby the slimes carrying the bullion have been collected as a wet mass. From this point, the methods of

treatment diverge considerably, which are divided into the following four groups—

- (a) Drying and smelting direct.
- (b) Drying, roasting, and smelting.
- (c) Acid treatment, drying and smelting.
- (d) Smelting direct with litharge and cupelling.

(a) This is a simple method, and consists in drying the slimes in shallow iron pans placed over or into any suitable furnace. The dried material is then broken to powder, mixed with flux, and smelted in plumbago crucibles. The bullion obtained is about 600 to 750 fine. The following proportions of flux have been largely used—

Dry slimes,	.	.	.	.	.	100 parts
Bi-carbonate of soda,	.	.	.	.	.	50 "
Borax,	.	.	.	.	.	35 "
Sand,	.	.	.	.	.	15 "
Nitre,	.	.	.	.	.	2-4 "

(b) In the second method the wet slimes are mixed with a little nitre placed on trays in a muffle furnace, and when dry the heat is increased sufficiently to oxidise a large portion of the zinc. The zinc oxide fumes given off are drawn away to a chimney by a carefully regulated draught. There are several types of furnace in use for this purpose, the most popular being the reverberatory introduced by Chas. Butters in the early days of the process, and the circular or rectangular coke type introduced more recently.

The oxidised charge may be smelted with the following flux—

Dry slimes,	.	.	.	.	.	100 parts
Bi-carbonate of soda,	.	.	.	.	.	40 "
Borax,	.	.	.	.	.	40 "
Silica,	.	.	.	.	.	15 "

The bullion by this method may be 800 fine.

A modification of the above was recommended by W. A. Caldercott \* in 1897. The slimes are dried nearly to dusting, then well mixed with silica and nitre in the following proportions—

Slimes,	.	.	.	.	.	100 parts
Sand,	.	.	.	.	.	30 "
Powdered nitre,	.	.	.	.	.	30 "

The mixture is spread on iron trays to a depth of about  $\frac{3}{4}$  inch thick and heated to a little over the boiling point of water. Then a live coal and a pinch of nitre is applied to the mixture, when combustion spreads throughout the whole layer and proceeds quietly, but intensely, giving off dense fumes of zinc oxide, and leaving behind a coherent non-dusting product.

The action is said to consist in the oxidation of a portion of the carbonaceous matter and base metals, and a partial combination with the sand to

\* *Jour. Soc. Chem. Ind.*, xvii. p. 3, 1897.

form a soluble silicate. The following is the flux used in smelting the product—

Oxidised slime,	.	.	.	100 parts
Anhydrous borax	.	.	.	40 „
Dry sodic carbonate,	.	.	.	10 „

This is said to give a good slag, very free from gold, and bullion about 800 fine.

In either of the above methods of eliminating zinc, there is a considerable risk of losing gold carried away with the zinc fumes and other causes. It is practically impossible to determine this loss with any accuracy, chiefly owing to the extreme difficulty of getting average samples of the material before and after treatment. Also, either method involves at least an extra handling of the material, which requires to be conducted with the utmost skill and care.

(c) The third method was mentioned by Butters and Clennell\* in 1892, and consists in adding sulphuric acid direct to the wet slimes in the clean-up tank, which dissolves out the zinc and other matter. The method is largely used in America, South Africa, and Australia.

T. H. Leggett† described the acid treatment as conducted at the Treasury mine, Witwatersrand. The clean-up vat is provided with a hood and flue, which can be readily raised and lowered. Concentrated sulphuric acid is added at intervals, about a bucketful at a time, until the whole of the zinc is dissolved. Next, the vat is filled with boiling water and the contents are agitated, when the hot liquor is drawn off and forced through a filter press. More hot water is added, and agitated until the whole of the slime has been drawn from the vat into the press. By forcing air through the press, the slime cakes are dried and are then ready for the smelter.

The following flux is recommended—

Slimes,	.	.	.	100 parts
Borax,	.	.	.	50 „
Bi-carbonate of soda,	.	.	.	30 „

The fineness of the bullion is stated as being 843. For each 100 lbs. of dried refined precipitate, 217 lbs. of sulphuric acid and 5800 lbs. of hot water were used.

The treatment at the Princes mine, Witwatersrand, as described by E. H. Johnson,‡ is briefly summarised as follows: The slimes are separated from the solution and washed with the use of a vacuum filter. They are then roughly weighed to determine the quantity of acid required. For each 100 lbs. of moist slimes, 100 lbs. of acid are used, and this is diluted to about a 10 per cent. acid. The slime is gradually fed by a hopper into the acid, which is continuously agitated by a stirring apparatus. When the action ceases the vat is filled with water and allowed to settle. The clear liquor is then syphoned off, water wash added, stirred, settled and decanted in succession,

\* *Eng. and Min. Jour.*, p. 365, 1892.

† *Trans. Inst. Min. and Met.*, vol. v. p. 147.

‡ *Eng. and Min. Jour.*, p. 220, 1899,—Paper read before Chem. and Met. Soc. S. Africa.

about four or five washings being sufficient. The slimes are finally dried on iron trays, mixed with the following flux, and smelted—

Dried slime,	.	.	.	100 parts
Borax,	.	.	.	66 "
Carbonate of soda,	.	.	.	9 "
Fluor spar,	.	.	.	9 "

The average fineness of the bullion during one year was 820. For 100 lbs. of dried original slimes, 133 lbs. of acid were used, and for 100 lbs. of dried refined slimes, 672 lbs. of acid were used.

An almost identical method was in use at the Kleinfontein mine, Witwatersrand, by F. C. Pengilly,\* the chief difference being that the slimes were roasted after the acid treatment,—the fineness of the bullion being 907.

**Treatment of Zinc-box Slimes containing Lead.**—The treatment of slimes so far described refer to those obtained by zinc shavings in their natural state, but since the introduction of the zinc-lead couple, special methods had to be devised for dealing with the precipitate.

The chief practical trouble arising from the use of lead acetate, was the difficulty experienced in the elimination of the lead from the gold bullion produced. According to T. L. Carter,† who first described the zinc-lead process as applied in slime treatment by W. K. Betty, several wet methods were first tried, such as the use of nitric acid on the precipitated sludge after the zinc had been dissolved by sulphuric acid. The nitric acid dissolved part of the lead, but not all of it. Hydrochloric and acetic acid have also been tried, with more or less success; the latter dissolved the lead almost completely, but proved expensive. The above authority recommends smelting with the following flux, to be used in crucibles with clay liners, presumably after sulphuric acid treatment—

Borax,	.	.	60 per cent.
Nitre,	.	.	19 "
Sand,	.	.	11·5 "
Soda,	.	.	7 "

It is stated that with this flux, bullion 876 fine can be obtained.

More recently, however, manganese dioxide has been used in the flux to oxidise the lead, clay liners being of course necessary in this case also.

After the slimes have been treated with acid to remove the zinc and calcined, E. H. Johnston and W. A. Caldercott‡ describe a method of employing manganese dioxide, as an oxidising agent when smelting, to oxidise lead and other base metals. They show that when nitre is employed for the same purpose it has a low efficiency, and give, as a probable reason, that the evolution of oxygen occurs at a temperature below that of combustion of the reducers.

\* *Eng. and Min. Jour.*, p. 340, 1898,—Paper read before Institute of Mining and Metallurgy, London.

† *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii. p. 142, 1898.

‡ *Ibid.*, vol. iii. pp. 21-24, 1902.

The following approximate proportions to flux slimes after acid treatment are recommended :—

Slimes,	.	.	.	.	100 parts
Fused borax,	.	.	.	.	20 to 35 „
Manganese dioxide,	.	.	.	.	20 to 40 „
Sand,	.	.	.	.	15 to 40 „

Data given of an actual smelting by F. Maxwell are as follows—

Slimes,	.	.	.	.	100 parts
Manganese dioxide,	.	.	.	.	21 „
Fused borax,	.	.	.	.	35 „
Sand,	.	.	.	.	35 „
Fluor spar,	.	.	.	.	5 „

The bullion obtained was 970 fine. Silica was present in the slimes to the extent of 7 per cent.

The total weight of calcined slimes dealt with was 202·8 lbs., producing 286 lbs. of slag and 1016·1 ozs. bullion.

It is pointed out that the use of manganese dioxide tends to enrich the slag in silver. The method is therefore not to be recommended for highly argentiferous slimes.

(d) However, all these methods of dealing with the precipitates from the zinc-lead couple, are at the present time being superseded, to some extent, by a very simple method, which consists essentially in smelting the sludge with litharge in a pan furnace, thus producing lead bullion which is afterwards cupelled. This extremely neat adaptation of the scorification assay method to work on a large scale, and to overcome a very real and serious difficulty, has been introduced and developed by P. S. Tavener, who has kindly placed at our disposal full particulars of his process, from which we have prepared the following condensed description.

The gold sludge is pumped from the clean-up vat to the filter press, and there made into cakes as usual. The fine zinc remaining in the vat is drained and collected, but is kept apart from the filter press cakes, both parts of the clean-up being separately dried in trays in an oven for not more than fifteen minutes, care being taken in each case that the mass is still coherent when removed from the oven. The slime cakes are immediately rubbed through a 4-mesh sieve, roughly weighed, and mixed with the flux. The mixture is then passed through a sieve and shovelled into the smelting furnace. Then the fine zinc is treated in the same manner and charged into the furnace above the slime, with the intention of preventing the latter from dusting.

It will be readily understood that the exact fluxing mixture to be used must vary with the composition of the sludge dealt with. This may be ascertained fairly well by small trials in the assay office, but generally speaking less slag is needed in the large furnace than in crucible tests. The quantity of litharge depends chiefly upon the amount of gold present, as it is said to be desirable to limit the value of the lead bullion produced to 8 or 10 per cent.



of fine gold. The following figures will give an approximate idea of the proportions hitherto found useful.

- (1) Slime, after partial drying, . . . 100 parts by weight  
 PbO, . . . . . 40 to 60 " " "  
 Assay slag, . . . . . 10 to 15 " " "  
 Slag previously used, . . . . . 10 to 15 " " "  
 Sawdust, . . . . .  $\frac{1}{2}$  to  $1\frac{1}{2}$  " " "  
 (2) Fine zinc partially dried, . . . 100 parts by weight  
 PbO, . . . . . 100 to 150 " " "  
 Mixed slag as above, . . . . . 25 to 30 " " "  
 No reducing agent except the zinc itself.

When the furnace charge is complete it is banked up towards the centre and lightly covered, first with litharge, secondly with readily fusible slag, and the fire may then be lighted. Two hours later, or thereabouts, the heat may be considerably increased, and a high temperature maintained. When the charge is molten, any by-products, such as sweepings, old slag, or anything that contains gold, is added to the bath and absorbed therein. Then the charge is well stirred, and sawdust is added to reduce any remaining litharge until the slag gives a clean sample on a rabble. The slag is next run off into pots in the usual manner as far as possible, and most of the remainder is waved off with the rabble. Then the fire door is opened and lime is thrown in to thicken the last of the slag, which is afterwards removed and kept for re-smelting next time. After a sample has been taken with a ladle, the clean lead is tapped and run into ingot moulds in the ordinary way.

The next step in the treatment is to cupel the lead bullion on a bone ash test, in which the usual cupellation practice is followed. When this operation is complete the gold is allowed to partially cool in the furnace until it is solid enough to be lifted up with a bar of iron. It is then broken in half and drawn out of the furnace. The cake is next broken up into smaller pieces of suitable size for re-melting in crucibles, so that it may be cast into bars.

This method has been developed into a practical success by P. S. Tavener at the Bonanza mine, Johannesburg, under the enlightened management of Francis Spencer, and the following figures show the actual amount of precipitate handled and quantity of gold obtained from July to October 1902.

	Troy. ozs.	Avoir. lbs.	Per cent.
Moist weight of filter press slime smelted, . . . . .	131,342	9,058	54
Moist weight of fine zinc smelted, . . . . .	111,104	7,662	46
Total weight of material treated, . . . . .	242,446	16,720	...
Lead bullion obtained and cupelled, . . . . .	221,407	15,269	...
Total amount of fine gold recovered, . . . . .	12,810	...	...

Hence we see that the average grade of the lead bullion during this period was nearly 5·8 per cent. fine gold.

One great advantage of this method is that it enables all sorts of sundry gold-bearing materials to be treated and realised, which otherwise gradually accumulate on a mine, and have finally to be sold to customs works for considerably less than the actual value of their gold contents. Some of these by-products have already been mentioned, and to these we may add old crucibles and liners left over from previous methods of refining.

Then, as we shall see in the chapter on working costs, this method is much cheaper than acid treatment followed by calcining and pot smelting.

In every comparative test that has been made so far, the new method has given a larger return of fine gold per pound of precipitate treated, the excess in some carefully conducted trials being over 10 per cent., thus confirming the suspicions long entertained that the losses incurred in calcining on a large scale were much greater than those shown by carefully conducted calcination tests in the laboratory.

The new method also prevents the loss of silver which occurs when manganese dioxide is used; and lastly, the danger of poisoning, which results from acid treatment of gold sludge obtained from arsenical ores, is avoided. For these last two reasons, it seems likely that the lead smelting method will be even more welcome and useful in other mining centres, than it is on the Rand goldfields.

**Smelting Operations.**—The object aimed at in fluxing gold slimes is to obtain a neutral slag, sufficiently fluid at a moderate temperature to allow the gold and silver particles to agglomerate and descend by gravity. To attain this end a rough analysis of the slime would be a great help to the smelter, as the actual proportions of fluxing materials are largely dependent on the composition of the slime. Much has to be found in an empirical way, and the smelter is often called on to use his judgment as to the fluidity, neutrality, and other factors. In this way two smelters may obtain equally good results, although the fluxes and their proportions may vary somewhat widely.

The proportions we have given in the preceding pages, are those which were in use at different stages of development of the cyanide process, and are merely to illustrate methods of procedure for special cases, rather than to be followed in a general way. The proportions of basic and acid oxides present in each case have to be taken into account, so as to select and adjust a flux that will produce a non-corrosive slag, as free as possible from shot metal, and at a reasonable cost.

The chief precautions to be taken in smelting are—

1. To dry the slimes and fluxes thoroughly before putting them into the melting pot.
2. To dry and anneal the crucible before use.
3. When charging the crucible while in the furnace, to stop the draught and use a funnel or hopper.
4. When firing, keep the crucible covered.
5. Heat and grease the ingot moulds before pouring the metal.

The furnaces usually consist of the ordinary square type used by assayers,

but large enough to take a No. 60 or 70 pot and with a stack of 30 feet or more. The top of the furnace is generally about on the same level as the floor, for convenience in lifting out the pots. Good oven coke gives perhaps the best results and is easy to manage, but lump coal has often to be used. In some districts charcoal is the only suitable fuel obtainable, and where this is the case a somewhat larger furnace than is used for coke becomes necessary. Producer gas has been successfully employed in the United States and elsewhere, but it requires more skill to deal with than coal or coke.

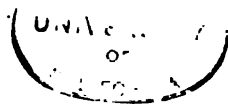
Nos. 50, 60, or 70 plumbago crucibles, with clay liners, are preferred in most cases. Their life may be taken at about 7 to 10 meltings.

An ordinary method of procedure in smelting is to first heat an annealed crucible slowly to redness, and then allow it to cool slowly. When cool enough to handle, fill with the mixture of slime and flux to within about an inch of the top. When the crucible is placed into the furnace it is gently worked down through the glowing coke until it rests on a firebrick, provided as a support. Fuel is then packed in all round the pot, and as the pot heats the mixture shrinks, when more is added. This is done preferably by use of a long wide necked funnel or hopper, and only a little at a time is dropped into the melting mass, until within about  $1\frac{1}{2}$  to 2 ins. of the top. Every precaution is taken to avoid dusting during the operations. When completely molten and all action ceases, the pot is withdrawn and the contents are poured into heated moulds. Some prefer to stir the charge vigorously before or after lifting it to pour, with the object of assisting fine shot metal to agglomerate and settle.

A certain amount of shot metal, however, remains disseminated throughout the slag, but for good work this ought not to exceed 0.5 per cent. of the total bullion. It is recovered by grinding the slag in a ball mill and treating the mass in an amalgamating barrel, or by treating with litharge and cupelling, as described under Tavener's method of treating slimes.

The floor of the smelting room is preferably covered with corrugated or fluted sheets of iron, No. 16 S.W.G. with corrugations of about  $1\frac{1}{2}$  in. pitch being suitable. The object of the corrugations is to enable dust and particles to fall into the valleys, and thus lessen the danger of bullion particles being carried away by the operators on their shoes.

There is always a certain amount of unavoidable loss in cleaning-up and smelting, and it is difficult, if not impracticable, to determine what this is. Some authorities put the minimum loss as low as 0.1 per cent. of the bullion smelted, while others maintain that 1 per cent. is nearer the mark. Tavener's method of dealing with gold slimes promises to bring the unavoidable loss to a minimum. This, however, can only be attained, in whatever method is adopted, by the employment of the most careful and skilled hands.



## CHAPTER XXV.

### APPLICATIONS OF THE CYANIDE PROCESS.

THE materials to which cyanide treatment is generally applied may be grouped into four classes, according to the means employed in its mechanical preparation for leaching purposes.

1. Accumulations of tailings from wet crushed ore.
2. Current production from wet mills.
3. Dry crushed ore.
4. Concentrates from vanners or similar machines.

**Tailings Accumulations.**—Formerly the most common method of collecting tailings was to allow the battery pulp, after amalgamation, to flow into comparatively shallow dams or reservoirs, formed by an excavation, or by building a dam wall across a natural hollow in the ground, and this system is still in operation in many mining districts.

The result is that at the point of inflow the heavy sand settles first, then the lighter and smaller particles, and finally the slime. As the filling continues, however, the layer of coarse sand not only grows in depth, but it also extends horizontally over the finer stuff which was deposited earlier, and the later fine sand also overlaps the previously deposited slime. Moreover, in order to fill all parts of the dam, the position of the inflow is changed from time to time, and the general result is a stratified mass with layers of sand and slime alternating, and these layers thin out to nothing and make or come in again in a most irregular manner. Although generally the upper part of the contents of the dam consists principally of sand, while the lowest parts are almost entirely slime, yet the greater portion of the dam is filled with layers of partially graded sand, interlaminated with well defined thin sheets of moist, coherent, plastic slime, but too thin and too numerous to be separated from the sand when the material is dug out for treatment. A partial separation can be effected by throwing the material against screens of about  $\frac{3}{4}$ -inch mesh, which allow the sand to pass through, but retain a large proportion of the slime lumps. This is, however, an expensive proceeding, and therefore unsuitable for low grade material, while for rich stuff it is not sufficiently effective.

The presence of these slime lumps is a decided disadvantage. If the lumps when put into the leaching vat contain as much water as they can absorb they are not only useless, but occupy space that might be otherwise filled with treatable sand; for, the saturated lumps cannot absorb solution and leach, and are therefore practically inert. Again, if the slime lumps are

partially or entirely dry when put into the vat, they are a cause of positive loss of gold, because the cyanide solution, as it passes on to the sand and dissolves gold, also soaks into the slime and carries dissolved gold with it. The solution thus absorbed cannot percolate out of the slime, and it is a well known fact that slime lumps which have gone into a vat in a plastic state, but nearly dry, have assayed sensibly higher after treatment than before. Therefore, either the slime lumps ought to be quite wet before cyanide solution is applied, which can be ensured by giving a preliminary water wash, or they must be dried, pulverised, and well mixed with a large proportion of sand before treatment. (For the effect of slime mixed with sand, see pages 50 and 54.)

The following means for fulfilling the latter condition have been used with some success. By ploughing and harrowing, followed by exposure to wind and sun, and finally shovelling through screens or passing through a Krupp disintegrator, the material from tailing dams can be brought into an improved condition for treatment when the original amount of neat slime does not exceed about 7 per cent. of the whole, but the cost of such preparation is somewhat high. If, however, the slime is excessive, the mixed material in the vat sinks down as soon as the liquor is applied, and packs together into a practically impervious mass, rendering satisfactory leaching impossible. We therefore recommend the following course of operations when dealing with poor tailings lying in dams.

Begin if possible with the clean sand at the top of the dam, and while the treatment of this is proceeding, the next portion, containing less than 7 per cent. of slime, can be ploughed and weathered as above described. The lower portion, containing still more slime, can then be put into the vats exactly as it is dug out of the dam, a first water wash being given to saturate the slime lumps. This method of treatment can then be continued as the excavation proceeds towards the most slimy part of the dam, so long as the operation continues to pay. The remainder must be treated, if at all, by close sizing and separating all the sand from the slime, or by treating the whole as slime, by the agitation methods described in Chapter XXXII.

In some instances the slime dams have been periodically emptied and their contents piled up into heaps. During this operation and while lying exposed, the slimes are to a large extent dried and pulverised, and carried away by the wind together with some of the finer sand, consequently such heaps are far more easily and successfully treated than stuff which has remained in the dams, although of course the quantity saved is considerably less.

In some parts of Australia and in Siberia it is customary to catch the coarser sands in small excavated pits, while the remainder of the pulp runs to waste or settles in a larger reservoir. The pits are cleaned out alternately as soon as each is filled, and the contents stacked in heaps. By this method a small portion of the total tailings is saved in an excellent condition for cyanide treatment, but the larger part, if saved at all, is deposited as a mixture of sand and slime, of such a nature that it is scarcely possible to treat it by any

means unless rich enough to pay for complete separation. This, therefore, is probably the worst of all methods of dealing with a battery product.

In addition to the losses indicated above and the mechanical difficulties described, there is a serious chemical disadvantage arising from the storage of tailings when pyritic ore is being crushed, namely, the partial oxidation of the pyrites, but this has been dealt with elsewhere. It is mentioned here as an additional objection to the above methods, and as one of the causes which led to improved systems of dealing with battery pulp for cyaniding purposes.

The plant used for treating such accumulations of sand is usually of the simplest description, and consists essentially of leaching vats, with a staging above for filling them, and, in large plants, a tunnel below for emptying, together with storage vats for solutions, and a shed containing precipitation apparatus and pumps. As an example of extreme simplicity, we may mention that in Queensland small plants have been erected (and worked profitably on 5 to 6 dwt. material) which consisted merely of two 10-ton shallow vats built on the ground-level, two solution vats, a hand pump, and a precipitation box. One man runs the material in a wheelbarrow up an inclined plank resting on the top edge of the vat staves and tips it into the vat, while a second man stands inside and spreads the material evenly. The vats are emptied by shovelling over the side and wheeling away in barrows. For large plants the types illustrated in figures 30 to 37 are suitable for treating accumulated sands. In figures 34 to 37 the sand heaps are indicated by dotted lines, and the collecting vats shown in these figures would in this case, of course, be omitted.

**Current production from mills crushing wet. Collecting Vats.**—The first device used for collection of pulp as it flows from the battery, with a view to immediate cyanide treatment, was a pair of shallow pits, each 120 feet long by 20 feet wide by 3 ft. 6 ins. deep, with masonry sides, and at the lower end a timber frame, with grooves to receive a series of wooden slats, each about 3 inches deep. The intention was to catch the sand and to allow the water and slime to flow over the slats. As the filling of the pit proceeded, the level of the overflow was raised from time to time by the addition of another 3-inch slat. A gentle downward slope of the bottom of the pit assisted the subsequent draining of the contents and their removal. When the pit was full, the flow of pulp was diverted to the other pit, and while this second one was filling, the first was drained and its contents were transferred in trucks to the leaching vats, and so the two receptacles were filled and emptied alternately. With pits of the length named all the slimes escaped, and unfortunately also much fine sand, while, if the length is much increased, layers of slime will occur as in tailings dams. But although this idea was abandoned in its first shape, it was developed in two most important directions. (a) The direct filling of treatment vats themselves, by providing them with overflow gates which could be raised as required, was introduced by Hennen Jennings. The pulp was delivered by a flexible hose pipe, controlled by hand. (b) The use of specially constructed intermediate collecting vats, as designed by Chas. Butters and

Captain Mein. The latter arrangement will be first described, as this is the most convenient order, but it may be remarked here that both methods have been combined in some of the latest practice on the Witwatersrand.

**Butters and Mein Collecting Vat for intermediate filling.**—This is of the usual construction of leaching vats, except that it has a circular launder surrounding its top. The vat is first filled with clean water, and the battery pulp is then delivered to it by means of a special distributor. The agitation caused at the surface of the water by the entrance of the streams of pulp is sufficient to prevent most of the fine slime from settling, and this consequently passes away with the water, which overflows around the whole periphery of the vat into the circular launder, and thence passes by suitable piping or fluming to a slime dam or the like. Meanwhile the grains of sand fall through the water in the vat in a continuous shower, and collecting at the bottom, gradually fill the whole vat with a mixture of fine and coarse sand. Some slime, however, remains in this collected material, and some fine sand escapes with the water. A filter cloth on a false bottom is provided for draining the collected material, and discharge doors for its removal.

A collecting vat is described in Chapters XXXIV. and XXXV. A distributor (see Chapter XXXV. and figs. 101 to 103), briefly described, consists of a conical receiving hopper, with several radial discharge pipes of various lengths and diameters, each pipe being bent horizontally at its outer end. This hopper is mounted over the centre of the tank and is free to revolve, the small power necessary being provided by the reaction of the water as it leaves the bends, the action being therefore entirely automatic. By this means the pulp is delivered in a number of concentric circles, with the object of attain-

TABLE XXXIV.

Number of Stamps, at 4 tons per stamp.	Diameter of Vat in feet.	Tons crushed per 24 hours.	Clear depth inside to hold 75 per cent. of tons crushed.
25	20	100	5' 0"
50	24	200	6' 6"
100	32	400	7' 6"

ing, as nearly as possible, an equal delivery over the whole area of settlement, because when the mass of collected material is of uniform texture throughout, the maximum amount of slime can be retained without rendering the product in parts unleachable. This in fact was the original purpose of the collecting vat, and is so still when used for intermediate filling, namely, to capture, for immediate transfer to the leaching vats, the greatest possible percentage of the solid material in the pulp that is consistent with efficient leaching.

To do this, it is necessary to have a proper proportion between the amount of pulp delivered and the diameter of the vat, for it is self-evident that a very small vat will chiefly retain the coarsest of the sand, while a very large one will catch too much of the slime. With suitable sizes it has been found that from 65 to 75 per cent. of the whole delivery can be collected as a leachable material, when crushing Rand blanket ore through 900-mesh screens. Under these conditions the sizes given in Table XXXIV. have given good results. To the figures of Butters and Smart \* we have added the third column, in order to make the table applicable to stamps of greater or less capacity.

The above sizes are not of universal application ; for very slimy ore much smaller sizes are necessary, while, on the other hand, with tailings from quartz rock, we have used 18 ft. vats to collect from 75 tons per day with good results.

As the separation of slime and sand takes place at or near the top of the water, it is clear that the depth of the vat can practically have no effect upon the nature of the material collected, so that this dimension is limited only by constructional reasons, or by the fall available on the proposed site of the plant. It is often convenient for systematic working that each vat should hold the material collected from one day's crushing, and the depths given in column 4 of the above table are quite sufficient for that purpose, and may be adopted in the absence of any other determining factor.

Allowing 24 hours for draining and 12 hours for discharging each vat, the minimum total capacity provided should be equal to three days' collection. That is to say, three vats are required if each collects for 24 hours, or two vats when each collects for 36 hours. But unless extreme economy in construction is absolutely necessary, it is better to provide a total capacity equal to four days' collection, so that any occasional delay in draining or discharging need not result either in loss of material or reduction of tonnage crushed.

When in any proposed plant there is a doubt as to the most suitable diameter for the intermediate collecting vats, it is better to have six of small diameter than three of a larger size, because if the vats are found to be too small for the whole of the battery pulp, this can be readily divided and delivered to two or more vats simultaneously ; whereas, on the other hand, if the vats should prove, when built, to be too large, there is no remedy except an increased delivery from the battery.

As already mentioned, the collecting vat must be filled with clean water each time it is used before the pulp is turned into it, so that the overflow may commence immediately. If the pulp is delivered into an empty vat it will take several hours to reach the top of the tank, and during that period the slime is settling as well as the sand. The filter cloth becomes covered with a sludgy layer of material, which is not only unsuitable for treatment, but prevents, or very much retards, the draining of the good sand above it, thus interfering with the regular and systematic working of the plant, as well as reducing the percentage of extraction.

\* *Proc. Inst. C.E.*, vol. cxx. part ii., 1895.



Also, if the battery stops during the filling of a vat, a good stream of water should be delivered continuously, to maintain the normal overflow until the battery is at work again, otherwise a thin impermeable layer of slime settles over the surface of the collected sand. When a stoppage has taken place, at least a dozen holes should be bored down through the material from top to bottom as soon as the filling is completed, to assist the draining of the sand which lies above the slime layer.

At the time when these collectors were introduced no practical means of working slimes had been perfected, and it was therefore desirable to allow no more slime to escape than was absolutely necessary for the proper treatment of the remaining material. And at the present time there are many mines in Australia and other countries where the conditions are not favourable to the use of a special slime plant, or a complete grading of the material. In such cases the need for treating as much slime as possible with the sand still exists, and under these conditions the use of collecting vats with immediate transference to treatment vats is undoubtedly the simplest and most practical system to employ.

Figures 30 and 31, where collecting vats are shown in dotted lines, and figures 34 to 37, illustrate the usual types of plant for this method of treatment.

**Collection and Treatment in one Vat.**—The method of treatment by delivering battery pulp directly into leaching vats was, as already mentioned, introduced by Hennen Jennings. In the early plants designed for this purpose the tanks were built of brick and covered inside and outside with cement. The bottoms of the tanks were level with the ground outside, and each had three openings which extend from top to bottom of the tank. These are used as overflow passages, and have a vertically grooved frame on each side, in which narrow strips of wood can be placed across the opening, one above another, as required. This arrangement for periodically raising the overflow, which is identical with that already described in reference to collecting pits, is now very largely used, and is generally known as a slat gate. A cast iron door, ingeniously suspended from wheels which travel on bearers placed above the opening, serves to close the vat during leaching. These overflow passages are made sufficiently wide for a truck to pass through, as they are also used for discharging the residues after treatment.

When a tank is to be filled, the door is left open and a few slats are placed in the bottom of the grooved frames. The pulp is then delivered into the tank through a flexible hose, by means of which a labourer distributes it evenly over the whole area. The overflow level is raised from time to time by the insertion of additional slats, and the distribution of pulp through the hose is continued until the tank is full of sand. The cast iron door is then shut and tightened up by bolts and nuts. The leaching is carried out in the usual way, and the tank is emptied by opening the doors and shovelling the residues into trucks. At the beginning of this operation the sand has to be thrown through the doorway, but when a cutting has been made a truck can enter the tank, and after a time several trucks can be filled at once.

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The principal objection to this method of filling tanks is, that the uniformity of the collected material depends upon the intelligence and fidelity of the man in charge of the hose, who if lazy or stupid may easily make either of two blunders. Firstly, he may produce an unleachable tankful of material by the retention of too much slime, or, secondly, he may cause the waste of a large quantity of treatable sand. Further, if the delivery end of the hose is allowed to lie too long in one spot, the material will be stratified instead of being uniformly mixed, thus causing irregular leaching.

One great advantage of the method is that the quality of the product is independent of the size of the tank, and that the grade of material collected can be regulated to some extent, if required, by varying the height of the overflow. In this method a preliminary filling of the tank with clean water is not necessary.

A considerable amount of supervision is, however, required, and this is a drawback where labour is dear. It may be here pointed out that this particular method of filling is not essential to the direct treatment of battery pulp in one vat, but that the automatic collecting vat, previously described, when properly proportioned to its work, has the advantage of filling without supervision, and is therefore suitable for direct treatment with even very low grade material, which must be worked cheaply or not at all.

At the South Star mine, Victoria, Australia, the tailings were collected in ordinary collecting vats with distributors and peripheral overflow, then leached in the same vats, and finally sluiced out by the stream of battery pulp passed through a hose and nozzle. By this means the treatment costs were reduced to a minimum; but there are obvious objections to it, especially as regards loss of treatable material during the discharge by sluicing. It is in some cases justifiable, on the ground that it is better to treat 50 per cent. of the total crushing at a profit than 70 or 80 per cent. at a loss. In other cases, as for instance where a large part of the values is in the fine sands and slime, it cannot be recommended, because a larger proportion of fine material can certainly be treated when it is transferred from one vat to another, either before or during its treatment. This is on account of the better mixing obtained by the transfer.

**The Use of Hydraulic Classifiers.**—The methods of treatment so far described in this section, have been successfully applied to pulp from ores which contain very little mineral, or in cases where the pyrites in the ore has been effectively removed by vanners or by any other method of close concentration. On the Witwatersrand goldfields a few years ago the use of Frue vanners was almost universal; but in the few cases where tailings were treated without proper concentration, it was found that the pyrites, if separated from the sand residues after treatment, was practically as rich as it was before treatment. Not, however, because the gold contents are unamenable to cyanide, but because a very much longer contact with the solution is required for pyrites than is necessary for the bulk of the tailings. To Henning Jennings and J. R. Williams belongs the credit of successfully applying

hydraulic classifiers for the purpose of separating the coarse pyrites together with a small quantity of coarse sand, so that a long treatment can be given to the portion thus separated, and the usual shorter treatment to the remainder.

For this purpose the classifier or spitzlutte should be so designed that the coarse sand collected with the pyrites should only be sufficient to dilute the latter, so to speak, to such an extent as to enable the solution to surround each pyritic particle, as it has been found that neat pyrites alone packs with the faces together into a somewhat coherent mass, and prevents the proper flow of solution past the gold and pyritic surfaces. Any excess of sand above this necessary quantity merely increases the bulk of material which has to receive long treatment, without giving any compensating advantage. In practice it is found that a coarse product containing from 30 to 35 per cent. of pyrites will give satisfactory results.

The use of the spitzlutte for the above purpose led Williams to the appreciation of a most important principle, which, although it had been previously suggested, had not been hitherto practically applied. This principle is, that the leaching of equal sized grains can be carried on much more expeditiously and effectually than is possible with a mixed mass of grains of all sizes. This is due to the following causes: firstly, that with the sized grains the proportion of voids in the mass is greater than with unsized grains, and consequently there is more room for the passage of solution around the particles, and probably less capillary action to retard its draining; secondly, the conditions being uniform throughout the mass of sized grains, the flow of solution will also be uniform in all parts. Therefore, if continuous leaching is adopted, each part of the solution is more completely displaced by the next following portion, while with intermittent leaching, each separate solution drains off more completely. With mixed sands the moisture retained when drained is generally from 10 to 20 per cent., while with the same grains sized it may be reduced to between 5 and 10 per cent. in the aggregate, according to the perfection of the sizing. Consequently, a less quantity of dissolved gold is left behind at each draining, and a higher total extraction is obtained with a given amount of washing with sized grains.

There is still another advantage arising from the use of classifiers, namely, that the more complete draining and the larger voids allow a more perfect penetration of air between the grains after each solution is run off, and therefore give a more effective supply of oxygen. When a number of grains of sand carrying gold are each coated with solution whose total surface is in contact with a plentiful supply of oxygen, we have all the necessary conditions for the rapid dissolution of the gold, although only for a short time.

Returning now to the work of the spitzlutte, it may be pointed out that so long as materials of different specific gravity are being dealt with, this apparatus does not separate the grains merely according to size. In fact, Rittinger,\* in speaking of the product from spitzluten, says, that "as the difference in volume will always be greater than the difference in weight, it

\* *Lehrbuch der Aufbereitungskunde*, 1867-73, p. 183.

may therefore be said that equal falling bodies are similar in weight rather than similar in size."

And in the first spitzlutte this is exactly what is required, namely, to get much smaller particles of pyrites than of sand included in the product. Then, having eliminated in the first operation nearly all the coarse and some of the fine material of high specific gravity, that is to say, the pyrites, the further separation with the following classifiers will take place more strictly according to the sizes of the grains, which again is just what is required for the purpose in view.

The action of a spitzlutte can be readily understood by those not already acquainted with it, by the diagrams, figs. 38 to 40.

Fig. 38 shows a vertical tube with a column of water flowing through it in an upward direction. If a number of grains of various sizes and specific gravities are dropped through the funnel *a* into the ascending water, it is evident that the rate of flow may be so adjusted that some of the larger and heavier grains will fall downwards and settle at the bottom of the tube at *b*, while the lighter and smaller grains will be carried upwards and discharged with the water. And for each rate of flow there will be a corresponding set of grains which will just fall. Similarly, if battery pulp can be sent in an upward stream at a certain velocity through a pipe, some of the grains will be left behind and some carried forward by the ascending current. In a tube such as that shown in fig. 39, for instance, the heavy coarse sand will fall back and collect at *b* in the bend. Such an apparatus would, however, soon choke itself, but fig. 40 shows a continuous arrangement in which there is a small outlet for coarse sand at the bottom of the bend. Naturally, some fine stuff and turbid water will flow out with the sand if no means be provided to prevent this. But by forcing in a small stream of clear water through pipe *c*, the downward pressure of the slimy water is overbalanced and the coarse sand escapes alone in the clear water by pipe *d*. By regulating the size of the openings *b*, *c*, and *d*, and varying the height of the discharge *e*, a clean product can be obtained, the nature of which, however, depends essentially upon the *velocity of flow* in the main tube. Therefore, by repeating the operation with a series of decreasing velocities, several grades of sorted material can be obtained.

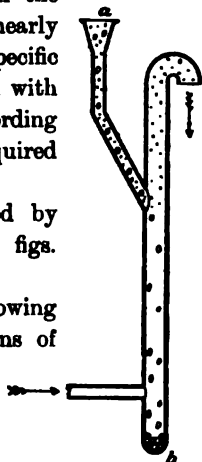


FIG. 38.—Action of the Spitzlutte.

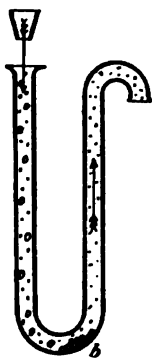


FIG. 39.—  
Action of the  
Spitzlutte.

Mixed grains can also be sorted by means of a horizontally flowing stream of water, for, by sufficiently reducing its velocity, it will be unable to carry forward the largest and heaviest particles, which will consequently sink to the bottom. A further reduction of velocity will cause other finer grains to sink,

until finally nothing but water flows away. This is the principle of the spitzkasten, which is merely a box, rectangular at the top, with sides and ends so inclined inwards that they come nearly to a point at the bottom. A small outlet is provided at the bottom for the escape of the falling grains. A series of such boxes, with successively increasing width, gives a set of roughly graded products. Practical forms and details of both kinds of apparatus, and rules for designing them, are given in Chapter XLI.

As a rule, very few grades of material are separated in practice. Usually one or two classes are made from the first 10 to 20 per cent. of the whole crushing, and then about 60 to 70 per cent. of the whole is treated in one lot. The remaining 20 to 25 per cent. is slime, from which, however, a small quantity of very fine sand is sometimes separated, this being either treated separately, or more commonly returned to the tailings launder or tailings wheel.

The following particulars of the Rand practice in 1897, as given by

E. D. Chester,\* are interesting for comparison with the classification methods used in the same district at the present time. Battery pulp, carrying about 6 dwts. per ton, passed first through a spitzlutte which took about 1 dwt. per ton in 17 per cent. of concentrates, the value of these being therefore about 11 dwts. per ton. The 5 dwts. overflow from the spitzlutte next passed through a spitzkasten, the underflow from which carried about 61 per cent. of the battery output at 5 dwts. 8 grns. per ton, and the overflow carried off 22 per cent. of slime, worth 4 dwts. per ton.

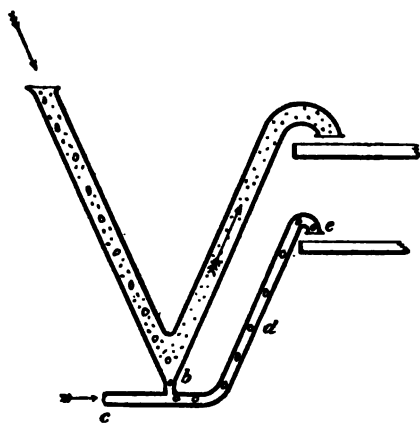


FIG. 40.—Spitzlutte.

The following details of the practice at the Bonanza mine, Johannesburg, at the present time, have been kindly supplied by P. S. Tavenor, the cyanide manager. The pulp, the value of which when leaving the battery is about 9.6 dwts. per ton, passes to a spitzlutte 20 feet wide, which separates 10 per cent. of concentrates worth about 25 dwts. per ton, and containing 30 per cent. of pyrites. This underflow product is delivered by a hose into treatment vats: 24 feet in diameter and 7 feet deep, fitted with slat gates. The overflow from the spitzlutte, containing then 7.9 dwts. per ton, is delivered into similar vats of the same size, which collect about 65 per cent. of the whole tonnage, with an average value of 9 dwts. per ton. The overflow from the gates of these sand vats passes to a spitzkasten 6 ft. wide  $\times$  3 ft. long  $\times$  3 ft. 6 ins. deep, which separates out a little fine sand, and this is returned to the tailings pump, to

\* "Gold Mining in S. Africa," *Min. Ind.*, vi. p. 385.

pass again through the classifiers. Lime is next added to the overflow by a special automatic machine designed by P. S. Tavenor (see page 213). The water then contains only slime, which amounts to about 25 per cent. of the ore crushed, and has a value of 5 dwts. per ton. This mixture passes to a large spitzkasten 31 ft. 6 ins. wide  $\times$  13 ft. 6 ins. long, in which the whole of the slime is settled, and 61 per cent. of the water is clarified and returned at once to the battery by gravitation. The slime settling tanks clarify another 36 per cent. of the water, which is pumped back to the battery, and only about 3 per cent. of the whole water used in crushing is left in the settled slimes, the further treatment of which is dealt with in a following chapter.

Another variation in classification now practised on the Rand may be briefly mentioned. In this case the products separated by the classifiers are: (1) Concentrates about 4 per cent.; (2) Coarse sands 9 per cent.; (3) Medium sands 67 per cent.; and (4) Slimes 20 per cent. These later figures compared with those of 1897 given above show that the tendency has been towards closer concentration, although, of course, the quantity of first product will depend partly upon the amount of pyrites in the ore as well as upon the degree of concentration attained.

The following figures from the annual reports of the Crown Reef G. M. Co. show how the percentage of leachable products has increased since the introduction of classifiers.

Year.	Per cent. Concentrates.	Per cent. Sands.	Total per cent. leachable material.
1896	4.12	69.71	73.83
1897	3.98	68.74	72.72
1898	3.31	71.79	75.10
1899	not given	not given	81.54

**Classification in New Zealand.**—At the Waihi mine methods similar to those just described have gradually displaced the process of dry crushing previously employed. At the Waihi mine, under the management of H. P. Barry, 95,829 tons were dry crushed and 63,496 tons were milled with water during the year 1901, and the rest of the stamps were then to be converted for wet crushing, which is found to be more suitable for the mineralised ore. At the Victoria Mill of this Company the stamp duty of dry and wet crushing respectively was 1.711 and 2.361 tons per stamp per day with 40-mesh screens. The average extraction for the year from wet crushed ore was, according to assay, 87.4 per cent. of the gold and 66 per cent. of the silver, while the actual recovery was 81.1 per cent. and 64.3 per cent. respectively. It may be noted that some of the coarser sands are separated and re-ground.



The following particulars of the classification plant have been supplied by Clement Dixon, and relate to the pulp from 100 stamps. The whole of the mill pulp is lifted by plunger pumps to a trestle launder 70 feet high, which carries it to a spitzkasten consisting of three compartments, each 3 feet square. The underflow with the sands goes to collecting vats 38 ft. by 8 ft., in which a preliminary treatment is given before transfer to a second vat.

The slime and water overflow, from the first spitzkasten, passes on to a second one 36 ft. square, divided into 25 compartments, each 7 ft. by 7 ft. at the top, the underflow from which contains practically all the slime, and is run into shallow settling vats, to be thickened before it is pumped up to the slimes plant.



## CHAPTER XXVI.

### DOUBLE TREATMENT.

ANOTHER modification of the cyanide process which has been introduced on the Rand, and which is very generally adopted there and on other goldfields, is to give a partial treatment in one vat, and then to transfer the material, after draining, to a second vat, where the leaching is completed. There are two advantages aimed at in this system of working: first, that in the turning over of the whole mass during transfer, any closely packed unleachable portions in the first vat may be broken up and distributed amongst the rest of the sand; and second, that the material already partly treated and moistened with cyanide solution may be exposed to contact with the oxygen of the air during the removal from one vat to the other.

Usually, in the first vat an alkaline wash, a weak cyanide solution, and a medium solution or part of the strong solution are passed through the material, in order that a sensible amount of the cyanide solution may be present at the time of transfer. Then, in the second vat the remainder of the strong solution is given, followed in the usual way by weak solution and water wash. There is no doubt whatever, as to the improvement in percolating qualities caused by the transfer from one vat to the other, in the case of ordinary mixed unclassified material; and even with the sand product of the present classification methods there is a sensible advantage in this respect, for it is found that these sands occupy a larger space, being sometimes 20 per cent. more in the second vat than in the first. And also as regards oxidation, there must be some slight advantage, particularly when dealing with unclassified sand.

But as the two processes—classification and double treatment—have the same objects in view, namely, improved percolation and oxidation, it is evident that, so far as these objects are attained by either process, it thereby renders the other less necessary. The question then naturally arises whether both will continue to be used together, as at present, or whether a further development of either method will render the other unnecessary?

If in double treatment the stuff could be completely broken up in presence of atmospheric oxygen during its passage from one vat to the other, either by revolving screens or by a disintegrator, the beneficial effects of transference would undoubtedly be increased, and for non-pyritic material the whole of the sands could probably in many cases be treated together, with a satisfactory extraction. But with Rand ore and many other pyritic ores, on account of the long treatment required by the larger pyrites particles, it would still be

necessary to carry out the present usual classification into three or more sizes. Therefore development in this direction does not seem likely to entirely displace classification.

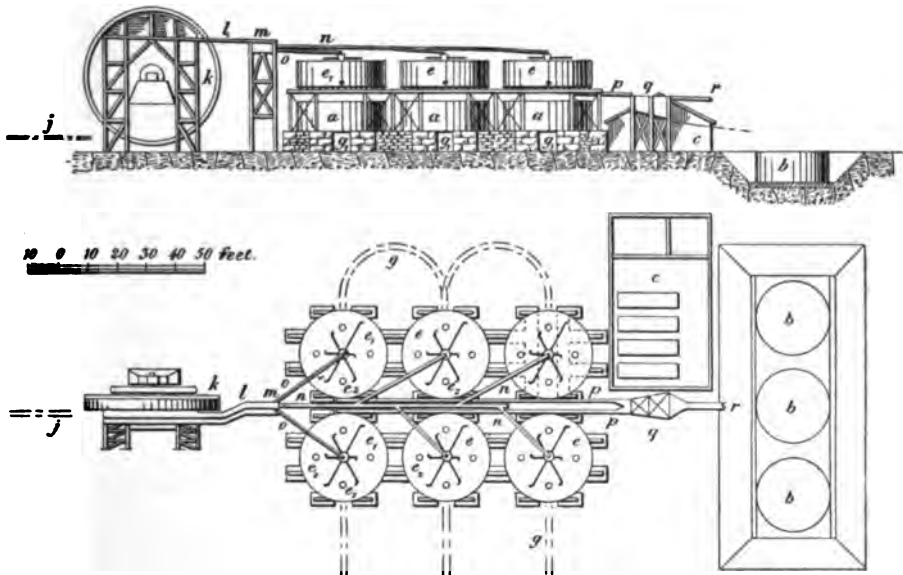
On the other hand, if the material can be really well sized into, say, six classes, then the method of leaching continuously with oxygenated solution, or intermittently, with percolation of air after each solution is drained off, should give a better oxidation than can be obtained by merely shovelling once from one vat to another, or by transference in trucks or with belt conveyors, and consequently double treatment could be dispensed with. But there are some difficulties in the way of development in this direction, for the spitzlutte and spitzkasten are not instruments of precision, so that it is not always possible to obtain six well sized products by their use. Then the cost of the extra classifiers and the extra water has to be considered, as against the somewhat smaller cost of a single treatment plant. Another point of difficulty is that by separating into many grades, the quantity of each produced in a given time will be small, so that in most cases either small vats must be used, or the long time occupied in filling them will necessitate a considerable increase in the total vat capacity of the plant. The latter difficulty may, however, be overdone to some extent, by mixing two or more sizes in the same vat. This, of course, partly defeats the object aimed at, but when we classify into only three sizes, there is a fine product between the sand and slime which is capable of being dealt with in the leaching vats, and which only goes to increase the quantity of material to be treated by the more expensive method of agitation. If, on the other hand, we classify into six sizes it becomes possible to separate this fine product, and thus diminish the bulk of slime. This is of especial importance where the slimes are too poor to treat.

**Treatment in three vats.**—It has been found in some cases that the partially classified sands, which form the bulk of the material, are not sufficiently uniform in texture to ensure the complete moistening of the whole mass with cyanide solution before transfer, and that consequently the exposure to the air during transfer does not greatly assist the dissolution of the gold, so that the final extraction is practically no better than when the transference takes place before treatment. Therefore in some cases double treatment has been abandoned and the earlier method resumed. But in other plants the difficulty has been met by using three vats in succession for each charge. That is to say, the first vat is used for collection only, and the collected sands are immediately drained and transferred to a second vat, where they receive the first part of the cyanide treatment, which is completed after removal to a third vat. It is clear that this principle is capable of indefinite repetition,—in fact, until the value of the increased extraction obtained by the last transfer is equal to or less than the cost of the transfer.

**Double-tier Plants.**—In recent plants erected specially for double treatment the vats are in two tiers, but it is as well to point out that this is not an essential feature of the process. In fact, it is no more necessary in this

case than it was in the older method of collecting in one vat and treating in a second one, provided that suitable precautions are taken against loss of solution when the first treatment vat is emptied. For example, figures 41 and 42 represent the usual type of double treatment plant, but the types shown in figures 34 to 37 are also quite suitable for this purpose.

The chief advantage of the double-tier arrangement is that the transfer can be effected at a smaller cost, but against this may be placed the disadvantage, that the contents of any upper vat can only be discharged into the



FIGS. 41, 42.—Double-tier Plant.

- |   |   |
|---|---|
| <i>j</i> Launder from battery.  | <i>e<sub>2</sub> e<sub>3</sub></i> Discharge doors of upper vats. |
| <i>k</i> Tailings wheel.  | <i>p</i> Launder for slat gates.                                  |
| <i>l</i> Launder from top of wheel.                                   | <i>q</i> Slime spitzkasten.                                       |
| <i>m</i> Spitzlutte separating concentrates.                          | <i>r</i> Launder to slime plant.                                  |
| <i>n n</i> Launder to sand vats.                                      | <i>a a</i> Second treatment vats.                                 |
| <i>o o</i> Launder to concentrate vats.                               | <i>b b</i> Solution vats.   |
| <i>e e</i> First treatment for sands with distributor and slat gates. | <i>c</i> Precipitation sheds.                                     |
| <i>e<sub>1</sub> e<sub>1</sub></i> Ditto for concentrates.            | <i>g</i> Tram line for discharging lower vats.                    |
|   | <i>g<sub>1</sub></i> Discharge tunnel for lower vats.             |

one immediately beneath it, so that any delay occurring with either means a loss of time for both vats. In the other types of plant, the material can be transferred from any vat of the first set into any one of the second series with equal facility. Of course, this objection to the double-tier plant can be obviated by allowing sufficient headroom for men with trucks between the two tiers, but this would add to the cost of construction, which in any case is much greater than for plants of the other types.

With regard to the lifting of the material, when the plant can be so arranged that the whole of the clarified water from sand vats and slime

spitzkasten can be gravitated back to the battery, so that the whole of the pulp has only to be lifted once by tailings wheel or pump, the advantage is rather in favour of the double-tier arrangement. For, in the other case on a flat site (figs. 36 and 37) the collecting vats have to be below the battery outflow, so that the sand has to be lifted to the second treatment vats and the water pumped up to the battery again, and the cost of these two operations separately is a little more than that of lifting the pulp at first. But where the site has a sufficient slope, so that the battery pulp can be gravitated to the collecting vats and the sand transferred to the second vats without lifting, then only the clarified water has to be raised, and therefore the double-tier plant under these circumstances shows no economy of handling material, sufficient to counterbalance the extra cost of its construction. Moreover, in merely shovelling from one vat to another, the stuff is only turned over once, while with transference by trucks or belt conveyor it is turned over twice.

**Double Treatment practice on the Rand.**—The following description of double treatment as carried on at the Bonanza mine, based on figures kindly given by P. S. Tavener, affords a good illustration of the most recent Rand practice, and may be conveniently read in connection with the particulars of classification at the same plant given on page 178.

The strong solution is made up to 0·2 per cent. KCy, once for each charge, and varies from this down to 0·04 per cent. The medium solution ranges from 0·1 to 0·08 per cent., and the weak solution carries about 0·03 per cent. KCy. Each vat holds 114 tons of material, this having been ascertained by actual weighing.

The concentrates, when drained in the upper vat, receive first 10 tons of weak solution, then 65 tons medium strength, after which they are again drained and transferred to the lower vat, this part of the treatment occupying five days. In the lower vat the charge receives 50 tons strong, followed by 75 tons medium and 65 tons of weak solution, after which it is drained and discharged. This occupies twelve days, which makes seventeen days for the whole treatment.

The sands receive 10 tons of weak solution and 20 tons of medium in the upper vat, and 50 tons of strong solution, 20 of medium, and 25 of weak wash in the lower vat. The time occupied is three days for the upper and five days in the lower vat.

**Multiple Treatment in America.**—The foregoing method of employing vats superimposed for double treatment is also in use in America, but this is likely to be superseded by a system now being introduced by Chas. Butters and H. W. Blaisdell. The vats are all arranged on the one level, as shown diagrammatically in fig. 44 in plan and elevation. Eight or any other suitable number of vats are placed in series, and the plant is extended by arranging parallel to these any number of similar rows.

When the ore is delivered from the mill as it is crushed, it first passes through a slime separator or classifiers in the usual way, and the underflow is then carried by launders to the collecting vats, where the pulp is delivered by

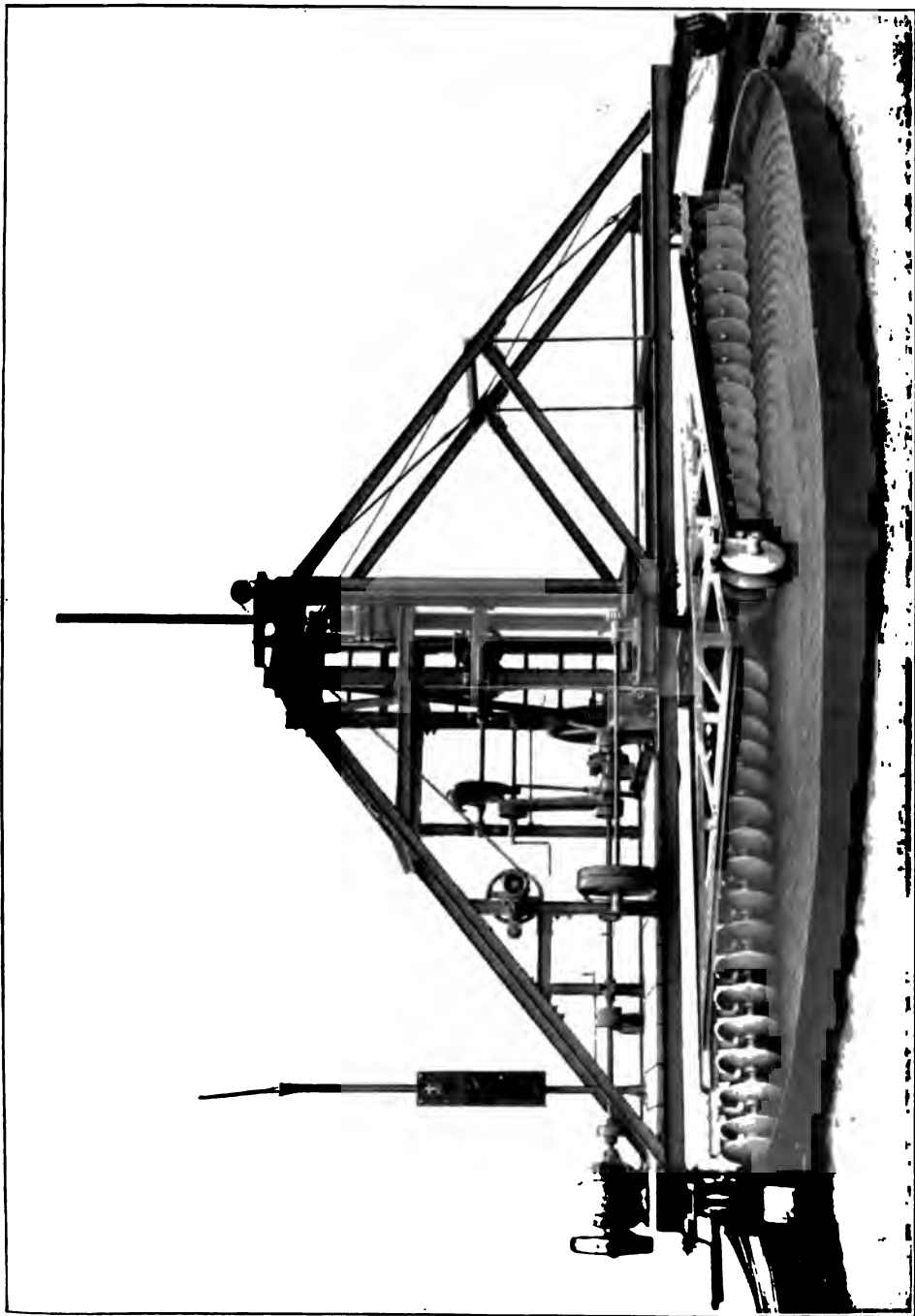
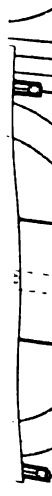


FIG. 43.—The Blaisdell Vat Excavator.



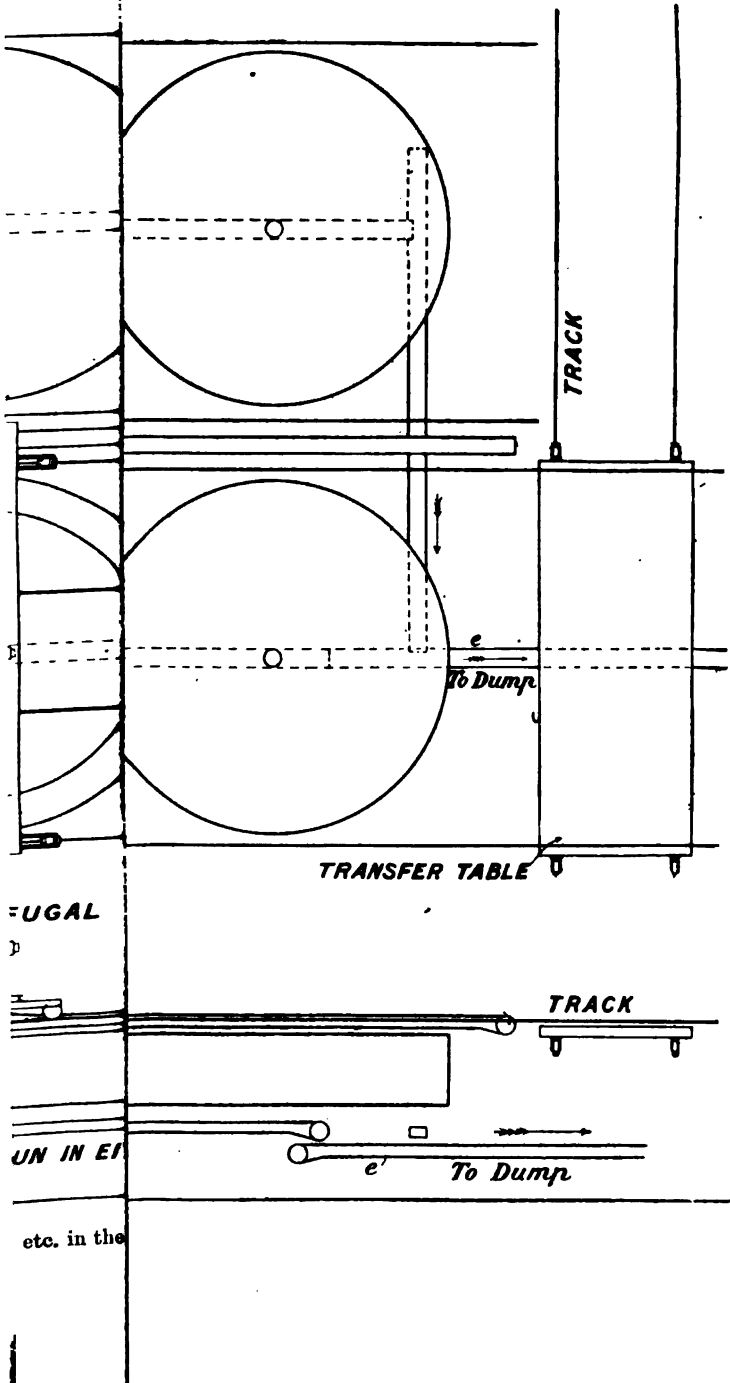
UG



UN

etc.

[To face fig. 43.







means of a Butters' Distributor. In these vats the ore gets its first cyanide treatment, and is then discharged on to a belt conveyor *a*, which delivers it to belt *b*, and thence to *c*, by which it is elevated and delivered to any other vat in the series. The ore is transferred from *c* by means of a movable tripper *d* to a Blaisdell Centrifugal Distributor, which breaks up adhering lumps of sand and distributes the charge evenly in the vat. The charge is then treated in the usual way, and is discharged again on to the belt conveyor *a*, from which, if desired, it may be transferred to any other vat in the series for a third treatment, or may be discharged and carried away to the dump by simply reversing the direction of the conveyor. The residue is then delivered to a conveyor *e*, which finally disposes of it to the dump. The belts are each 16 in. wide.

The vats are discharged through single central discharge doors, and a special machine has been devised for this purpose, known as Blaisdell's Vat Excavator. This is shown in fig. 43, taken from a photograph. It consists essentially of a number of steel rotating discs, similar to those largely used in cultivators, attached to arms, which are rotated from the centre by means of an electric motor. The discs are arranged at an angle with the arm as shown in plan, fig. 44, and when the arms are rotated in the direction shown by the arrows the discs push the sand inwards to the centre, where it falls down a hole previously made, to the belt conveyor beneath. It is obvious that the tendency of the discs is to cause the sand to build up as it approaches the centre, and to counterbalance this action two short arm-carrying discs are arranged at right angles to the long one. This machine is said to have a capacity of 100 tons an hour working in a 30 feet vat, and requires 7 horse-power.

The Butters distributor, the excavator, and the Blaisdell centrifugal distributor are all carried on carriages that run on a track over the whole length of each series of vats. In order to transfer these parts to any other series of vats, a transfer table is provided, running on a track at right angles to the main track at the end of the series, as shown in fig. 44.

At the works of Chas. Butters and Co. Ltd., Virginia city, Nevada, the Butters distributor is supported by a crane which stands outside the line of vats, but midway between the two collecting vats. This plant is also arranged so that the ore can be brought in direct from accumulated heaps of tailings by means of the conveyor *f*, and then any vat in any series can be charged direct by means of a Blaisdell centrifugal distributor.

Chas. Butters is also introducing a special machine for drying and aerating sands and slimes, known as Blaisdell's Aerator. It is similar to that shown in fig. 45, page 195, except that the sprayer is dispensed with, and the shelf *a* and the bottom of the machine are covered with perforated plates, leaving hollows into which compressed air is forced, which rises upward through the sand as it is being turned over by the discs. This machine is used to oxidise reducing matter in old tailings and slimes before being charged into the leaching vats, and it is also proposed to use it to aerate ore as it is being transferred from one tank to another in the double treatment.

## CHAPTER XXVII.

### DIRECT TREATMENT OF DRY CRUSHED ORE.

IN this application of the cyanide process, the first trouble met with results from the necessity of drying all ore which contains more than about 2 per cent. of moisture, before it can be crushed dry. Drying is done in kilns or furnaces, and this often partially decomposes pyrites and other sulphides, arsenides, tellurides, etc., forming compounds that increase the consumption of cyanide, and render the gold and silver less soluble.

In the earliest New Zealand practice the ore was dried in circular pits sunk in the ground, each of these being about 20 feet in diameter and capable of holding 100 tons of ore and 25 tons of firewood, the ore and fuel being placed in alternating layers. The bottom of the pit is formed by a brick arch with a discharge door over a tunnel, through which the dried ore is removed in trucks, in lots of 50 tons at a time. After each withdrawal of half the charge the kiln is filled up again with layers of ore and wood, so that the method is practically a continuous one.

At the Crown mine at Karangahake, when dry crushing was practised there, the ore was dried in a furnace having a firebox, over which is a steeply inclined rectangular flue provided with inclined iron shelves, so arranged that they prevent the ore from touching the back and front walls of the furnace, and allow space for the products of combustion to pass up around the lumps of ore. It is similar in principle to the well known Stetefeldt furnace. The Rothwell furnace is used for drying in Colorado and other parts of America. It consists essentially of a tapered revolving cylinder, usually about 20 feet long, 4 feet in diameter at the receiving end, and 5 ft. at the delivery end. There is a brick firebox at the large end, and a brick dust chamber and chimney at the small end. The cylinder, of  $\frac{1}{2}$  in. sheet steel, is divided into quadrants by two longitudinal partitions of  $\frac{3}{8}$  in. steel plates, which cross each other at right angles and extend nearly the full length of the cylinder.\* The motion and inclination of the cylinder cause the ore to pass slowly along towards the firebox end, and it is dried by the heated gases passing in the opposite direction. Many of the well known types of roasting furnaces are also used for drying ore in various districts, as for instance the White-Howell and the Argall furnace in the United States and West Australia.

But with most American ores the amount of metallic mineral is so great that merely drying the ore renders the subsequent treatment difficult, if not

\* *Engineering*, July 1, 1898.

often impracticable, on account of the partial decomposition already referred to, and this is especially the case with ores containing much arsenical pyrites and certain tellurides. In these cases when the ore is dried before crushing, it is often advisable to roast after crushing, and when roasting becomes necessary, it is usually found most economical to dead roast.

New Zealand practice illustrates the use of stamp mills for dry crushing, while American plants show what is done with Krom rolls and similar machines. Ball mills have been used on a large scale in Chile and Western Australia, and Griffin mills chiefly in the latter place. It will be convenient to describe the methods practised in these countries in the above order.

**New Zealand practice with Stamps.**—It is generally admitted nowadays that a stamp mill working dry will crush through a given mesh, only about half the quantity that is delivered by the same mill when crushing wet with a suitable supply of water. But it does not follow that the ore is crushed equally fine in both cases. In fact it is almost self-evident that the fineness of the crushing for any given material will be dependent upon the time it remains in the mortar box, and in some cases a dry stamp mill will be nearly as efficient as a wet one, because in the latter case a much finer screen is required for a fine product than in the former. For instance, at the Waihi Mill, where the ore is a very tough amorphous quartz, a 900 lb. stamp falling 7 ins., at 92 drops per minute, crushes about 1.55 tons per stamp in 24 hours through a 40-mesh screen. The fact that 80 per cent. of the product from this tough ore will pass an 80-mesh screen, shows that the material is not discharged as soon as it is reduced to the screen size, and therefore receives much extra pounding before it is light enough to be blown or drawn through the screen. At Waihi this is an advantage, because it is necessary there to crush very fine in order to liberate the gold, and with the dry mill this can be done without the use of a very fine, and consequently easily breakable screen. Similar results are obtained at other mines in the Hauraki peninsula. In 1901, with slightly heavier stamps, the duty was raised at the Waihi to 1.7 tons per stamp.\* One great practical objection to stamps for dry crushing lies in the large quantity of dust which permeates the whole atmosphere inside the mill, and which constitutes a serious danger to the workmen, from its irritating effect upon their lungs.

The ore when crushed is usually carried by screw conveyors to storage bins, from whence it is delivered in trucks to the treatment tanks. Sometimes the section of tram line over each vat has a lateral traversing motion, so that a truck upon it can be tipped over any part of the vat. In the most recent plants a complete system of belt conveyors is used for charging. The entire treatment is given in one vat, and is similar to the single treatment method already described, but the vats are only filled to a depth of 2 ft. 6 ins. on account of the presence of a quantity of fine material. A vacuum is used to assist the leaching of the second and following solutions, the suction being applied in the following way: A steel cylinder is connected by a pipe, provided with a cock,

\* Annual Report issued April 11, 1902.

to the leaching vat below the filter. This cock being closed, the cylinder is exhausted by a vacuum pump to 20 or 24 ins. of mercury; it is then closed to the pump and opened to the vat, so that the suction draws the solution into the cylinder. When leaching is completed, the cock connected to the vat is closed, and the solution is run out of the cylinder, which is then again exhausted by the pump. This indirect method of applying the vacuum, instead of connecting the pump directly to the vat, is adopted to avoid injury to the pump valves by any grit remaining in the filtered solution.

It is well known that when dry ore is treated, a larger quantity of wash water can be used than in the case of wet crushed ore, without increasing the bulk of solution in use. In this district the following figures represent the average practice: For each ton of ore, 0.35 ton strong solution, 0.3 ton of weak solution, and 0.35 ton of water wash.

In 1901, according to the annual report, the extractions at the two dry crushing mills of the Waihi Company were—

*Waihi Mill.*—Assay extraction, 87.7 per cent. of the gold and 48.4 per cent. of the silver.

Actual recovery, 86.3 per cent. gold and 51.7 per cent. silver.

*Victoria Mill.*—Assay extraction, 89.4 per cent. of the gold and 50.6 per cent. of the silver.

Actual recovery, 88 per cent. gold and 52.5 per cent. silver.

These results are obtained with ore from the upper levels of the mine, but, owing principally to the increase of metallic mineral matter in the ore in the lower levels, dry crushing in this district is gradually being displaced by a modified form of wet crushing, which will be referred to later.

**American practice with Rolls.**—In the United States rolls of the Krom and other types have been used almost exclusively for the dry crushing of ores for cyanide treatment. Some very large and well equipped plants have been erected for this purpose, and we have selected the Golden Gate Mill at Mercur, Utah, U.S.A., as a good typical example for description, both as regards design and treatment.

For many of the following details we are indebted to articles by W. R. Ingalls and by L. Janin jr. in volumes vii. and viii. respectively of *Mineral Industry*.

The works are built on eight levels excavated on a steep hillside; the total fall from the grizzlies to the discharge from leaching vats is 130 feet. The ore is raised in skips to the top level by means of an inclined tram 800 feet long.

It is dumped over grizzlies, the fines going direct to a storage bin, while the coarse stuff passes through a 6-inch Gates Crusher to the same bin. There are three classes of ore, known as oxidised, talcose or mixed, and base, for each of which a separate storage bin is provided. The material is carried from these bins by belt conveyors over straight line dryers, which are used when necessary. The dry ore is sent to Berthelot sizing screens, from which the fines pass at once to the finishing rolls, while the remainder goes through the coarse crushing rolls and over a second vertical screen to the finishing rolls.

The following sized screens are used, namely,  $\frac{3}{8}$ -inch mesh for oxidised ore,  $\frac{1}{2}$ -inch for mixed ore, and  $\frac{1}{8}$ -inch for base ore. The three classes of ore are still kept separate after passing the finishing rolls, as their subsequent treatment is different. The fully oxidised ore is ready for the leaching vats, but the mixed ore is carried by belt conveyor to two Holthoff-Wethey Calciners, 100 feet long by 12 feet wide each, with a capacity of 145 tons per 24 hours, and a consumption of 8 tons of coal. The base ore is elevated to four Brown-Jackling straight line furnaces, 100 feet long by 12 feet, each burning 7 tons of coal and roasting 70 tons of ore per day.

Here we have examples of practically all the methods of furnace treatment preparatory to cyaniding, that is to say, drying before crushing when necessary; calcining after crushing, in order to drive off the combined water from the talcose ores, to improve their leaching qualities; and lastly, complete roasting for the arsenical ores, which contain before roasting 1 to  $2\frac{1}{2}$  per cent. of arsenic and 2 to 5 per cent. of sulphur, and from 0.1 to 0.15 per cent. of arsenic afterwards.

When charging the leaching tanks, a layer of oxidised ore is spread over the bottom to a depth of about 10 inches, and the remainder of the tank is filled with a mixture of all three kinds. The ore is trammed to the tanks, which are 50 ft.  $\times$  25 ft.  $\times$  5 ft. deep, in side tipping trucks, three men charging a tank with 260 tons in 8 hours.

The following cyanide treatment is given. A strong solution containing 0.4 per cent. KCy is admitted under the filter cloth at a pressure of 14 feet of head, and rises to the top of the charge in 8 hours. It remains on the ore for 16 hours, after which it is allowed to percolate downwards, and fresh strong solution is added at the top as required for 24 hours. This is followed by a 0.3 per cent. solution, then weak solution and water washes are applied, about 2 tons of liquor altogether being used per ton of ore treated.

Another large plant has been erected by D. C. Jackling at the Republic Mill, Washington,\* in which the general arrangements are similar to those of the Golden Gate, but in this case the ore passes through a sampling mill before going to the main crushing plant. In this sampling department the ore passes from the storage bin to a No. 5 Gates Crusher, thence by an elevator to the first revolving trommel. The fines go to a set of Gates high grade rolls, and the oversize returns through an H Gates Crusher to the first elevator, and so on until all can pass the trommel screen. After the ore has passed the rolls, a first sample is taken by a Brunton Automatic Sampler, the rejections going by elevator No. 2 to the sampled ore storage bins. The sample is lifted by a third elevator to a small set of rolls, and then passes through two sampling machines in succession. The sample from the second of these amounts to about 1 per cent. of the original weight of the ore, and is further reduced in quantity by hand sampling, and reduced in size by the small crushing apparatus usually found in large assay laboratories. All the ore from the sampling mill will pass a half-inch screen.

\* *Eng. and Min. Journal*, vol. lxx. p. 638, 1900.

**Chilian practice with Ball mills.**—The following account of a large installation of Ball mills at the Atacama Mineral Co., Chile, is condensed from a paper read by Sidney H. Loram before the American Institution of Mining Engineers.\*

The ore is broken in a Blake Crusher to pass a 3-inch ring, and is then delivered to eight No. 4 Grusonwerk ball mills, each of which contains 1320 lbs. of balls and requires 11 h.p. at 25 revolutions per minute. The output has reached  $6\frac{1}{2}$  tons in 24 hours, but over a long period, including stoppages and repairs, has only averaged  $5\frac{1}{2}$  tons per mill per day when crushing through an 80-mesh screen. Usually 60 per cent. of the product obtained under these conditions will pass a 100-mesh screen. One man can hand-feed the eight mills. The amount of steel abraded from balls and plates is 4.27 lbs. per ton of ore, in addition to which 1.42 lbs. is thrown away in worn out parts, so that the total loss amounts to 5.69 lbs. of steel per ton of ore crushed. These figures refer to the special malleable steel supplied by the makers of the machines.

This exceedingly fine grinding is necessary because the gold is extremely fine, and is said to be so thoroughly distributed through the gangue that even minute particles of the latter may sometimes completely enclose the gold. The ore is described as being very hard and tough, and is therefore, according to our experience, of the kind least suitable for these machines. Moreover, in most mining districts it is very seldom necessary to crush so fine as in this case, so that we may consider the above results to represent the work of a ball mill under the hardest conditions which are likely to occur in practice. See Table XXXV. page 193.

The ore, after crushing, is elevated in trucks to an overhead tram line and tipped into the leaching vats, where it receives an ordinary single treatment with the following quantities of solution per ton :—

Alkaline wash,	0.22 ton containing	0.01 per cent. KCy.			
Strong solution,	0.44	„	„	0.30	„ „ „
Weak	„	0.22	„	„	0.1 „ „ „
Water wash,	0.11	„	„	„	0.01 „ „ „

The 0.01 per cent. KCy, of course, is the strength at which the water wash comes off from the vat.

One point of interest in connection with this plant is, that sea water was used throughout without any ill effects, except that at starting “a rather heavy precipitate was formed of magnesium hydrate and carbonate,” which, however, was hardly noticeable with the small additional quantities of sea water required in the subsequent working.

But our principal reason for giving the above particulars is, that they afford a very fair comparison of ball mill work with the stamp mill crushing in New Zealand, for, according to S. H. Loram's description of the Atacama ore, it appears to be physically very similar to that of the Hauraki peninsula.

\* *Trans.*, vol. xxix. pp. 497-502.

Thus, as regards fineness, we have—

Ball mill ...	Crushed 80-mesh ...	60 per cent. passes 100-mesh screen.
Stamps ...	40 " ... 80 " "	80 " "
Stamps* ...	80 " ... 77 " "	90 " "

A careful consideration of the above figures leads to the conclusion that the products are equally fine in both cases. The ball mill takes 2 h.p. per ton crushed in twenty-four hours, and we estimate the power required for the battery, whose results are given, at 1·98 h.p. for the daily output of 1·55 tons, which is equal to 1·28 h.p. per ton per day. The labour for driving and feeding should be the same in either case, but the wear and loss of metal with ball mills is at least twice as much as it is for stamps. As far as our comparison goes, therefore, the stamp mill is a cheaper dry crusher than the ball mill for very tough ore requiring very fine grinding, but against this is the advantage that the dust difficulty can be readily obviated in the case of the latter machine, and the mill atmosphere kept clear and wholesome.

**Ball Mills and Griffin Mills in West Australia.**—In treating the oxidised ore on the Kalgurli goldfields, wet crushing followed by classification and double treatment, as on the Rand, has been the general rule, but the necessity for giving a dead roast before treatment of sulphide ores, has led to the introduction of dry crushing for this latter class of material. The ore is usually soft and contains a large amount of the silicates of alumina and magnesia, and consequently, with any method of crushing, a large quantity of fine slime is produced; but as the gold in the ore is itself extremely fine, this ready sliming of the crushed material cannot be looked upon as a disadvantage, but rather as a favourable circumstance which reduces the cost of milling. In fact, at some of the mines the whole of the ore is intentionally reduced to slime before treatment by cyanide, in order to get the highest possible extraction.

It is evident, therefore, that the direct treatment of the whole dry product by percolation, as in New Zealand and America, is impossible in this district, so the usual practice is to deliver the crushed and roasted ore to a mechanical mixer, in which it is made into a pulp either with water or cyanide solution. This pulp is then passed through classifiers to separate it into coarse and fine sands and slime.

The following details of the work carried on at the Kalgurli gold mines and the Lake View Consols are abstracted from a pamphlet by Robert Allen, M.A., B.Sc., which was published by the W. Australian Government.

At the first named plant the ore passes over a 2-inch grizzly to a No. C Comet Crusher, breaking to  $2\frac{1}{2}$  inches. When necessary, the ore passes through a White-Howell Dryer, and thence to six No. 5 Krupp ball mills. The mill product gives the following results by screening:—

Passing 120-mesh ...	40 per cent.
„ 80 „ ...	30 „ „
„ 85 „ ...	30 „ „

\* John M'Connell, *Trans. Inst. Min. and Met.*, vol. vii. p. 27.



The last named screen is used on the mills. A 45-inch Sturtevant fan draws the fine dust from the ball mills to cyclone settlers, whence it is sent to the furnaces to be roasted with the main product, these furnaces being provided with large dust chambers. The roasted stuff is carried to a mixer and pulped with water, thence to spitzkasten, which separates sands from slime. The sands pass over copper tables, then to Halley concentrators, and finally to leaching vats, where they undergo treatment for thirty days. The strong solutions are continually circulated by an air jet, which also aerates them. About two-thirds of the whole crushing is treated by percolation in this way. The concentrates from the Halley tables are amalgamated and slimed in Wheeler pans, and treated by agitation like the rest of the slimes.

The general features of the methods used at the sulphide plant of the Lake View Consols mine are similar to those just described, but instead of spitzkasten for separating sand from slime, an ingenious modification of the collecting vat is used. This vat has a revolving stirrer, which can be raised and lowered. The whole of the roasted ore is delivered to this vat together with a cyanide solution, and the speed of the stirrer is so adjusted that the slime is kept in suspension and overflows with the solution, while the sand settles in the vat. The sand thus collected is afterwards discharged into a pit, from whence it is delivered to the treatment vats. In this plant only about one-third of the total ore crushed is treated by percolation.

At those mines where the whole of the ore is reduced to slime when ball mills are used, their product after roasting is subjected to further reduction, either by grinding dry in a tube mill or wet in Wheeler pans. The tube mill has been used for some years in cement works, but as it has only recently been applied in mining, a brief description of it is advisable. It consists of a steel cylinder revolving on an axis slightly inclined to the horizontal, and containing a number of pieces of broken flint. The product of the ball mill is fed into the upper end of the cylinder, and is reduced to a very fine powder by the grinding action of the flints as it slowly passes between them to the discharging end. These mills are also used in a modified form for wet grinding. The Wheeler Pan is too well known to need description. Where Griffin mills have been installed this additional grinding plant is not used, for the Griffin mills with 15-mesh screens give about the same degree of fineness as the ball mill with 40-mesh. Where sliming is required it appears, therefore, that the first named machine has some advantages, but with ordinary quartz and quartzose ores the free delivery of the ball mill which causes the above difference is a distinct advantage, in that it tends to a uniform size of grain, determined by the mesh of the screen. As a comparatively coarse but uniform product is the chief desideratum in the large majority of cases, it is likely that the use of ball mills will extend in the future for dry crushing. With quartz, a ball mill may be expected to give a product of which from 80 to 90 per cent. is retained on an 80-mesh screen, while with Rand banket under the same conditions about 60 to 70 per cent. will be retained.

In Tables XXXV. and XXXVI. we have collected a number of figures given

TABLE XXXV.—Results obtained with Ball Mills.

Name of Plant.	Description of Mill.	Mesh of screen.	Tons crushed in 24 hours.	H.P.	H.P. for each ton crushed per day.	Authority.	Nature of material crushed.
Commonwealth, Arizona,	Jenisch No. 5	40	23	12	0·52	<i>Eng. &amp; Min. Jour.</i> lxx. p. 74.	Not stated.
Mt. Morgan, Queensland,	Krupp "	20	23	13	0·57	N. F. White.	Soft oxidised ore.
" "	" "	35	19	10	0·53	" "	" "
Atacama, Chile,	" "	80	5·5	11	2·00	S. H. Loran.	Compact tough quartz. Crystallisation microscopic or absent.
" "	" "	30	8	11	1·38	O. W. Lee.	Blue unoxidised rock.
Associated gold mines of W.A.,	" "	30	22	15	0·68	Robt. Allen.	Sulphide ore.
Lake View Consols, W.A.,	" "	40	25	15	0·60	" "	" "
" "	" "	40	40	25	0·63	" "	" "
Kalgurli gold mines, "	" "	35	25	15	0·60	" "	" "

TABLE XXXVI.—Results obtained with Griffin Mills.

Name of Plant.	Mesh of screen.	Tons crushed in 24 hours.	H.P.	H.P. for each ton crushed per day.	Authority.	Nature of material crushed.
Gt. Boulder Persaverance, W.A.,	15	40	25	0·63	Robert Allen.	Kalgurli sulphide ore.
Gt. Boulder Proprietary, W.A.,	15	26	25	0·96	" "	" "
De La Mar, Nevada,	...	32	25	0·78	<i>Min. Ind.</i> , viii. p. 308.	Not stated.

by various authorities, independent of the makers, which show actual results obtained in practical work with these two machines. It will be seen that, under the same conditions as to material and product, each requires about the same power. The slower speed at which the ball mill works is a practical point in its favour, as this means cheaper foundations and less noise and vibration in working.

The following figures given by N. F. White\* are of great interest, as affording a direct comparison of results obtained with rolls and ball mills on the same class of soft oxidised ore, through screens of the same size. The power given in each case includes the driving of dryers, elevators, screens, and mills.

TABLE XXXVII.—*Comparative Power required for Rolls and Ball Mills.*

Mills and Machinery.	Screen.	Tons crushed per day.	H. P.	H.P. per ton crushed per day.	Tons crushed per day for the H.P.
2 stone breakers, dryers, elevators, 8 sets of rolls, screens, countershafting.	20-mesh	125	100	0·80	1·25
4 stone breakers, 4 dryers, 4 elevators, 16 No. 5 ball mills, countershafting.	20-mesh	369	321	0·87	1·15

As regards power, therefore, the rolls have a slight advantage, but this is more than counterbalanced by the extra labour required in attending to the separate screens and elevators which return the oversize from the several sets of rolls, whereas with the ball mill this operation of returning the oversize is continuously performed inside the machine itself.

**Other examples of Dry Crushing and Cyaniding.**—At the George and May mine near Johannesburg, a quantity of banket ore was successfully treated by simply passing it through a single Gates Crusher and leaching directly in vats. The extraction was reported at 70 per cent., being about equal to that crushed through a battery screen.

We have met with oxidised ore in Gippsland, Australia, which gave the following results on a small scale after crushing to pass a 30-mesh, and in another case to pass only a  $\frac{3}{8}$ -inch screen :—

Product passing 30-mesh gave 73·3 per cent. in 7 days.

„ „  $\frac{3}{8}$ -inch „ 67·2 „ „

These results were as good as those obtained in working on a large scale with the same ore after crushing in the battery.

Other similar instances have been noted, but they are exceptional cases, where the ore was originally so heavily mineralised that the subsequent

\* *Trans. Inst. Min. and Met.*, vol. vii. p. 50.

complete oxidation of the mineral makes the ore not only friable, but also

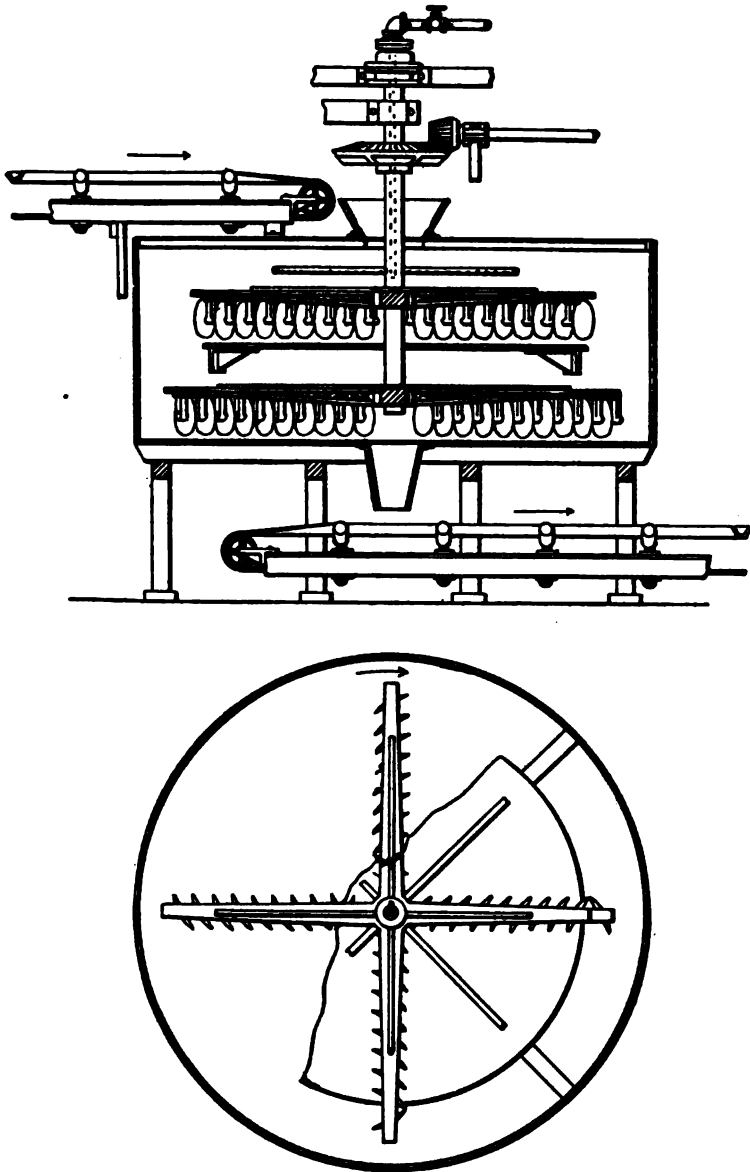


FIG. 45.—The Blaisdell Mixer for moistening pulverised ore with cyanide solution ; sectional elevation and plan.

*porous* ; consequently the solution can not only readily penetrate *into* the coarser pieces and dissolve the gold, but can also freely drain *out* again.

At the Lisbon-Berlyn and other mines in the Transvaal outside the Rand district, dry crushing has also been adopted. According to Matthew T. Brown only  $1\frac{1}{2}$  dwts. per ton could be obtained by wet crushing and amalgamation at this mine, while 40 per cent. of slime was produced. The ore is now passed successively through a Blake rock breaker, two Marsden fine crushers, and finally a set of Gates rolls, with screening at each stage of its crushing. The average extraction for twelve months on 10 dwts. ore has been 68 per cent. from the direct cyaniding of this material.

**Moistening Dry Crushed Ore.**—A special apparatus has recently been devised and used in the United States for moistening dry crushed ore, known as Blaisdell's Mixer, and is shown in fig. 45 in plan and sectional elevation. Its object is to prepare the ore in order that it may be conveyed and distributed in the leaching vats without danger of loss from dusting, and further, to improve the texture of the ore for leaching purposes and aerate it while moistened with cyanide solution. It consists essentially of two sets of arms, keyed on a vertical suspended shaft, which carry a number of discs, arranged at angles similar to those in the excavator already described, page 185. The upper arms are caused to rotate within a covered circular chamber over a shelf *a*, and the lower ones over the bottom of the chamber. The ore is fed through the hopper *b* and falls on to the shelf *a*, where it gets turned over and over again by the rotating discs, and at the same time is sprinkled with a cyanide solution. This is done by a perforated pipe *c*, which is fed through a hollow in the shaft. The upper set of discs are arranged to throw the ore outwards, mixing it at the same time, when it falls over the edge on to the bottom of the chamber, where the lower discs push it inwards to a central discharge. It then falls on to a belt conveyor to be carried to the leaching vats. A 12-ft. machine is said to have a capacity of 100 to 200 tons per hour, and requires 5 horse-power.

## CHAPTER XXVIII.

### CRUSHING WITH CYANIDE SOLUTION.

IN the year 1893 this method of treatment was introduced by the African Gold Recovery at the old May battery near Johannesburg. The mortar boxes were supplied with solution, and the pulp, after amalgamation on the plates, was led directly to the leaching vats. The solution when partially cleared of slime was returned to the battery. This process was abandoned after a short trial, but the cause of failure was not officially stated. It is likely, however, as no method for treating slimes was then known, that the difficulty lay in that direction. For if the greater part of the slime which overflowed from the leaching vat was returned to the mill, it would soon accumulate to an unworkable extent. If, on the other hand, the slime was merely settled and thrown away, the consequent loss of some gold and cyanide solution would sufficiently account for the non-success of the experiment.

A very successful application of the same principle, but with important modifications, is now in use in New Zealand, where it was first introduced at the Crown mines, Karangahake, by John M'Connell, in 1897. The nature of the ore in this district has already been described on page 187.

One important variation from ordinary wet milling practice is the small amount of solution used in the battery, namely, from  $1\frac{1}{2}$  to 2 tons for each ton of ore crushed. This limited supply has two advantages, the first being that the extremely fine crushing which is required can be effected without using a very fine screen. The following comparison in Table XXXVIII. of sizing tests given by J. M'Connell \* illustrates this point. In both cases the ore was crushed through a 40-mesh screen.

TABLE XXXVIII.

		With 4 tons of Solution per ton of Ore crushed.	With 2 tons of Solution per ton of Ore crushed.
Retained on 60-mesh,		13.5 per cent.	3 per cent.
"    "    90    "		20.5    "	21    "
Passed	90    "	66.0    "	76    "

\* "Notes on Dry and Wet Crushing with Cyanide Treatment in New Zealand," *Trans. Inst. of Min. and Met.*, vol. vii. pp. 26 *et seq.*

The following figures in Table XXXIX. relating to the same ore crushed wet and dry through a 30-mesh screen, given by the same authority, show that practically identical results can be obtained by the two methods.

TABLE XXXIX.

Retained on 40-mesh,	Dry crushed Ore.		Wet crushed with 1·6 tons of Solution.	
	1·20 per cent.		1·00 per cent.	
„ „ 60 „	9·00	„	8·00	„
„ „ 90 „	12·56	„	16·80	„
Passed 90 „	77·24	„	74·20	„

The second advantage above referred to, is that the volume of gold-bearing solution to precipitate from is kept within convenient limits, but is nevertheless nearly double the quantity used in the treatment of the dry crushed ore.

Lime is fed into the mortar box with the ore to keep the solution alkaline. The pulp passes over amalgamated plates as usual and thence by launder to collecting vats with revolving distributors, in which the bulk of the material settles, while the overflow is passed through two settling vats in series, for deposition of the slime remaining in the water. The solution is then sufficiently clarified to pass through the precipitation boxes, after which it again goes through the mill.

More lime is added in the collecting vat when required, and when sufficient material has been deposited, the supply of pulp is diverted to another vat, while the contents of the first are allowed to settle. Then the clear solution is decanted from the top, the rest of the solution is drawn off through the filter with the help of a vacuum cylinder, and the dissolved gold remaining in the ore is washed out in the ordinary manner. The above description of the method is condensed from M'Connell's paper already referred to.

The advantages of this method, as against the previous system of dry crushing applied to the same ore, may be fairly stated as follows:—

1. A slight increase of stamp duty from 1·13 to 1·38 tons per stamp per day.
2. Absence of fine dust from the atmosphere around the mill and treatment plant.
3. Recovery of coarse gold, if any. Of course, with dry crushing, this could be done by amalgamating the residues.
4. Agitation of pulp in battery and launders.

5. Saving of cost of preliminary drying, and absence of partially decomposed mineral due to irregular heating.

6. Saving of wear and tear of conveyors.

The disadvantages are—

1. Increased volume of solution to precipitate from, as mentioned above.

2. Increased wear and tear of battery screens.

3. Necessity for separate treatment of slime product.

In this case, therefore, the balance is in favour of the later method, but in view of its possible application in other districts, it is necessary to point out one serious drawback, in the case of ores which are distinctly 'acid' when delivered from the mine, namely, that no preliminary alkaline treatment can be given to the material before coming into contact with the cyanide solution. Also, as compared with the ordinary methods of wet crushing with water and the after-treatment with cyanide, it has the enormous disadvantage of reducing the capacity of the mill to a very great extent, and of producing a finer grade product than is usually economical. It may, however, find a suitable application where very fine reduction is required, and where the slimes are filter pressed.

#### TREATMENT OF CONCENTRATES.

The cyanide process is now applied in many cases to materials which, a few years ago, would without question have been treated by roasting and subsequent chlorination. This remark applies especially to the products of close concentration, which are often cyanided raw, but, if rich enough, are preferably roasted. The cyanide process was at first applied to these refractory materials only, and were then treated by agitation, alternating with percolation. From a purely technical point of view, this is undoubtedly the quickest and most efficient method of extracting the gold from such material, whether raw or roasted. But the expense of continuous or repeated agitation of coarse particles of high specific gravity, limits the application of this method to rich material only.

Professor R. W. Lodge, of Boston, Mass., published\* some interesting results of experiments by W. A. Tucker bearing upon this point. The tests were made upon Frue vanner concentrates, containing about 66.5 per cent. of arsenical pyrites, some iron pyrites, galena and chalcopyrite, that assayed over 6 ozs. per ton. The solution used contained one per cent. of KCy. We have averaged and summarised the results in the following table (XL).

The experiments were made on small samples of only 25 grammes each, which explains the very evident discrepancies in the KCy consumption, but they nevertheless establish the advantage of agitation for that particular material in its raw state.

\* *Trans. Am. Inst. Min. Eng.*, vol. xxv. p. 90.



Similar experiments by John M'Connell \* on concentrates containing 0.5

TABLE XL.

Percolation.			Agitation.		
Time of Treatment.	Extraction per cent.	KCy consumed lbs. per ton.	Time of treatment.	Extraction per cent.	KCy consumed lbs. per ton.
16 hours.	58.69	5.16	2 hours.	87.28	2.48
71 „	60.45	9.84	5½ „	91.49	4.64
118 „	76.93	9.52	23 „	91.90	2.96

oz. gold and 13.9 ozs. silver, crushed to pass a 90-mesh sieve, give very similar results, thus—

TABLE XLI.

By Percolation.					By Agitation.				
Time in hours.	Per cent. KCy before.	Per cent. KCy after.	Extraction.		Time in hours.	Per cent. KCy before.	Per cent. KCy after.	Extraction.	
			Gold.	Silver.				Gold.	Silver.
70	1.2	1.0	70	23.4	16	1.2	1.1	90	83.8
70	0.6	0.45	64	12.5	16	0.6	0.49	82	66.5

On the Rand goldfields fairly large quantities of concentrates have been treated raw from time to time, and it has been found that the extraction is materially improved by turning the material over every few days during the treatment.

From this it would appear that one of the difficulties in the way of treating such material is mechanical rather than chemical, and is probably due to the tendency of raw concentrates to pack so tightly and cohesively, that the solution cannot get free access to all the gold. In some cases, therefore, it is good practice to reduce the whole of the material to a very fine state, and to treat it as slime by agitation and filter pressing. And in cases where the quantity to be treated is too small to justify the provision of expensive plant, the use of revolving barrels for raw treatment is advantageous.

But in the great majority of cases, a dead roast followed by ordinary

\* *Trans. Inst. Min. and Met.*, vol. vii. p. 32.

leaching in vats is to be recommended. The following figures obtained in treating concentrates containing arsenical pyrites and iron pyrites, and about 2 ozs. of gold per ton, will illustrate the effect of roasting in producing a more rapid and complete extraction.

TABLE XLII.

Hours under Solution.	Extraction Raw.	Extraction Roasted.
96	55 per cent.	69 per cent.
150	59 „	86 „
190	62 „	90 „

In both cases the figures indicate that a still better extraction could be obtained by longer treatment, but with the unroasted stuff, even if the rate of dissolution during the last interval were maintained throughout, it would require about twenty-four days' total treatment to reach the 90 per cent. that is attained in eight days with the roasted material. In the case of highly mineralised ores of values greater than 2 ozs. per ton, it must not be forgotten that the supply of oxygen, even in well aerated solutions, is often insufficient for complete dissolution of the gold in the time allowable. Also, that with the comparatively coarse gold often present in concentrates, local exhaustion of the liquor around the particles of gold is likely to occur, in respect of cyanide, at the gold surface, and oxygen at the pyritic surface. It is therefore useless to allow the solution to remain at rest for any length of time, and the best treatment is to maintain a continuous percolation with solution of medium strength, say, from 0.1 to 0.2 per cent., until the limit of economical extraction is reached. When the roasting is properly performed, the rest of the treatment is usually very simple, because the furnace product being thoroughly oxidised, the consumption of KCy should be small, while the physical condition of the material, being dehydrated and taking a granular form, is conducive to satisfactory leaching. This method is applicable also to concentrates containing sensible amounts of carbonate of lime, barium, etc., which prevent the application of chlorine, owing to the great waste of that solvent. In the furnace, these are, for the most part, converted into the corresponding caustic hydrates, which are not merely innocuous, but actually advantageous in the cyanide treatment.

**Preliminary Treatment with Acid.**—It often happens that pyritic material, either by oxidising in the mine, or by weathering after concentration, or even through imperfect roasting, may become partially decomposed, and several suggestions have been made with a view to the preparation of such material for cyaniding, without the expense of a complete roast. Some interesting work has been done in this direction by A. F. Crosse and W. Bettel, and from

their statements of their respective experimental results\* we extract the following brief summary.

Assuming that the free acid and soluble acid salts have been dealt with in the usual manner, the principal cyanide remaining in the Rand concentrates is the basic ferric sulphate, which being insoluble in water, is not removed by the ordinary preliminary water wash. Crosse found in one case 0.495 per cent. of iron existing in this condition. His view is that this ferric sulphate is decomposed on adding cyanide solution, forming hydric cyanide and ferric hydrate, and he also states that the freshly precipitated ferric hydrate forms ferro- or ferri-cyanide of potassium. He proposes to overcome the difficulty by dissolving out the iron by means of a weak solution of  $H_2SO_4$  or  $HCl$ ; and gives the following example of the results of this treatment:—

Concentrates in natural state.—Consumption of KCy 2.84 grammes, and extraction 51.8 per cent.

Concentrates after digestion with weak  $HCl$ .—Consumption of KCy 0.66 grammes, and extraction 84.3 per cent.

The advantages of this method are indorsed by W. Bettel, who worked independently on the same lines and arrived at similar results. His views are, however, slightly different as to the action of the ferric sulphate, which, he says, with cyanide of potassium will form ferricyanide of potassium, hydric cyanide, and ferric hydrate, but that the *latter*, when free from basic salts, is entirely unattacked by KCy of moderate strength. Therefore he says, that any method which will completely convert all the basic ferric salts into ferric hydrate and an inert salt, will prevent much loss in cyanide. For instance, sodium dioxide in excess appears to oxidise pyrites to ferric hydrate at once, without forming the intermediate ferrous state, sodic sulphate being also formed. He does not, however, propose to use sodium dioxide for this purpose in practical treatment, because the cost of it would be too great.

Crosse also found that by heating the concentrates to a temperature just under dull red, sulphurous acid is given off, and the insoluble ferric sulphate is thereby converted into ferrous sulphate and ferric oxide. It is still necessary to add a little  $H_2SO_4$  to the first two or three water washes, to prevent the formation of the basic ferric sulphate from the ferrous salt in the tank. By this method he obtained 88 per cent. extraction, from concentrates assaying about 3 ozs. per ton, in fourteen days. One objection to this appears to be the difficulty, when working on a large scale, of exactly regulating the temperature, with the consequent possibility that some of the stuff might be much worse after heating than before. However, it is quite possible that some of these experiments might have been worked out into practical shape, had not the method of rough separation of the pyrites from the battery pulp by spitzlitten given such good results, that for the time being it has superseded all other methods on nearly all the mines of the Rand. The treatment given to the product so obtained has already been described.

\* *Proc. Chem. and Met. Soc. of S. Africa*, vol. i. p. 98 *et seq.*

## CHAPTER XXIX.

### SLIMES.

MUCH has been done since Graham made his classic researches on the properties of fine solid particles held in suspension in liquids. That eminent scientist distinguished and classified substances that exhibit little or no chemical affinity, and apparently dissolve in solutions without any limit, which do not crystallise, and show little or no tendency to diffuse or subside, while in many cases they pass through animal membranes. He called these substances colloids. They are known to coagulate and settle on the addition of electrolytes, and form simple suspensions or heterogeneous mixtures. Since Graham's time it has been proved that colloids and fine sand or slime in water have many common properties; and in absence of proof that colloids are solutions, it is probably better to follow the more modern investigators, and treat the so-called colloid solutions as simple suspensions of finely divided insoluble substances.

**Cause of Slimes not Settling.**—Fine solid particles suspended in water exhibit a number of properties which force us to recognise that the particles are charged electrostatically, and when small enough are capable of repelling each other to such an extent as to counterbalance the action of gravity, becoming uniformly distributed throughout the liquid, and thus are prevented from settling. There are other causes of less importance which need not be gone into here.

**The Settling of Slimes.**—It has long been known that silica, clay, and other finely divided suspensions in water are readily caused to settle or coagulate on the addition of an electrolyte, the most active electrolytes being acids and salts of the heavy metals, even when in extremely dilute solutions. Lime water, for example, precipitates clay suspensions, although only 1 part of lime is dissolved in 500 parts of water.

The quantitative precipitation of slimes by electrolytes has been investigated by Linder and Picton, Schultze, Hardy and others, who have come to the conclusion that the coagulating power depends almost wholly on the valency of the positive ion, having practically the same value for any positive ion of the same valency, irrespective of the nature of the negative ion. If we put the value of a univalent positive ion at 1, a bivalent ion would be about 30 to 40, and a trivalent ion 500 to 1000. One part of a ferrous salt will completely clear 2000 parts of slimy water, while one part of a ferric salt is sufficient for 60,000 parts of the same water. One part of lime will clear from 1500 to 1800 parts of water, and magnesia has about the same value, but is much slower in its action.

**Why Electrolytes cause Slimes to Settle.**—Recent investigators have put forward the theory that the suspended particles carry statical charges, being either positive or negative, and repel each other, as already stated. The addition of a salt capable of dissociating produce ions, which it is assumed are negatively and positively charged, that neutralise the static charges on the slime particles. The slime particles are all positively or all negatively charged with respect to the water, but on the addition of an electrolyte, ions are introduced that carry both positive and negative charges, and these have the effect of neutralising the action of the static charges on the slime particles. Thus the positive ions make the slime particles less negative and the negative ions make the solution less positive, or *vice versa*, until the state is reached that the slime particles no longer repel each other, but become free to be carried down by gravity, or, as we say, subside or settle.

Spring \* and Van Bemmelen † have made some interesting investigations on this subject, which led them to believe that they proved to some extent the existence of static charges on silica, bone black, and precipitated colloids.

#### TREATMENT OF SLIMES.

Several definitions of 'slime' have been suggested from time to time, but it is not easy to find a really satisfactory one. The conception of slimes as regards their treatment by cyanide depends chiefly upon their physical properties; therefore the term 'slime' may be sufficiently defined for our present purpose by saying that it includes all that part of a crushed material which, after being mixed with water and allowed to settle for forty-eight hours, retains at least 40 per cent. of moisture and exhibits plastic qualities.

The composition of slimes varies very considerably, according to the nature of the ore from which they are produced. The following analyses by Dr Loevy, ‡ of Johannesburg, will show the kind of material which is being successfully dealt with on the Rand. Sample A was from the Jubilee and Salisbury mine, B from the Meyer and Charlton, and C from the City and Suburban.

	A.	B.	C.
SiO <sub>2</sub> . . . .	72·15 ...	74·51 ...	69·80
Al <sub>2</sub> O <sub>3</sub> . . . .	16·47 ...	14·08 ...	17·95
Fe <sub>2</sub> O <sub>3</sub> . . . .	4·42 ...	5·15 ...	3·75
MgO . . . .	1·26 ...	1·48 ...	...
Ca <sub>3</sub> (P <sub>2</sub> O <sub>5</sub> ) <sub>2</sub> . . . .	0·04 ...	0·07 ...	...
S (total) . . . .	0·70 ...	0·78 ...	...
SO <sub>3</sub> . . . .	0·03 ...	0·02 ...	...
Alkalies . . . .	0·05 ...	0·08 ...	...
H <sub>2</sub> O (combined) . . . .	3·40 ...	2·86 ...	...

From these figures he calculates that the probable ratio between silica and clay is 51 to 35.

\* *Rec. trav. chim. Pays-Bas*, xix. 204, 1900.

† *Ztschr. anorg. Chim.*, xxiii. 321, 1900.

‡ *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii. No. 5, p. 80.

The quantity of slime produced depends upon the nature of the material crushed, the method of crushing, and the size of screen used. Talcose ores naturally give the greatest proportion of slime relatively to the amount of sand, and crystalline quartz probably the least amount. In a previous chapter we have seen that anything which retards the delivery of the crushed ore from the crushing machine, tends to increase the proportion of slime produced. The following examples show the results obtained with several screens of different mesh when crushing Rand banket.

At the Langlaagte Royal mine\*—

With 1200 holes per square inch, 22·8 per cent. of slime.

900	"	"	"	17·5	"	"
750	"	"	"	14·3	"	"

See also Crushing to Cyanide, page 40.

Table XLIII. shows the variations in the amount of slime treated relatively to the sand at some of the leading mines on the Witwatersrand.

TABLE XLIII.—*Proportion of Sand and Slime treated on the Witwatersrand from official returns for September 1902.*

Name of Mine.	Tons milled.	Sand.		Slime.	
		Tons.	Per cent.	Tons.	Per cent.
City and Suburban, . . .	8,450	6,577	77·8	1,873	22·2
Crown Reef, . . . . .	13,624	10,676	78·4	2,808	20·6
Crown Deep, . . . . .	11,255	7,620	67·7	3,234	28·7
Durban Roodepoort Deep, . .	6,770	4,940	73·0	1,721	25·4
Ferreira Deep, . . . . .	4,972	3,600	72·4	1,340	27·0
Geldenhuis Estate, . . . .	8,875	6,884	77·6	2,565	28·9
Geldenhuis Deep, . . . . .	17,440	11,790	67·6	4,382	25·1
Guisberg, . . . . .	6,951	5,000	71·9	1,600	23·0
Jumpers Deep, . . . . .	9,224	6,515	70·6	2,985	32·4
Langlaagte Deep, . . . . .	11,341	8,838	77·9	2,180	19·2
Nourse Deep, . . . . .	7,576	4,867	64·2	2,156	28·5
New Primrose, . . . . .	11,484	8,102	70·6	2,717	23·7
Robinson Deep, . . . . .	11,306	6,512	57·6	3,701	32·7
Simmer and Jack, . . . . .	18,125	14,181	78·2	4,091	22·6
Village Main Reef, . . . .	11,570	9,075	78·4	2,830	24·5

*Note.*—The small but obvious discrepancies between tons milled and tons treated are due to the difficulty of obtaining complete accuracy of measurement.

As with sand, so with slime, there are three usual conditions in which it presents itself for treatment:—

- 1st. Accumulated in dams.
- 2nd. In battery water, after the elimination of all the sands by the methods previously described.
- 3rd. In a dry state, as separated from sand by blowing apparatus.

\* *S. African Min. Jour.*, Apr. 6, 1895.

In the early days of the cyanide process some slimes were treated by making them up into bricks which were burnt and crushed. The burning destroys the plasticity, and the crushed bricks are therefore leachable, but the method proved costly. The treatment of slime by mixing with sand has been referred to previously.

Nowadays, the gold is always dissolved by agitation with cyanide solution, and it is then extracted either by successive washings and decantations or by the use of filter presses.

We shall first deal with the collection of slime from battery water by subsidence; secondly, with the methods of dissolving gold by agitation; thirdly, with the question of the separation of the gold-bearing solution from the slime; and lastly, with the various practical applications in different mining districts.

**Decantation from Slimes.**—When ore is in such a fine state of division that the solution will not percolate through it by gravity, or only very slowly, it is often found advisable to treat it by agitating in the cyanide solution, and then, by virtue of the difference of specific gravities, the slimy particles may be allowed to settle, while the (lighter) solution containing the precious metals dissolved, may be syphoned or drawn off in a more or less clear state. Anything like a complete separation of the solution is not possible by this method, as much remains behind in the settled slimes. The slimes may be again agitated with fresh solution or water, and again settled and the clear liquid drawn off, but it is obvious that in practice this treatment must be limited, on account of an unwieldy quantity of solution accumulating, and the cost. The difficulty may be got over to some extent, by causing the slimes to coagulate and settle into the smallest possible bulk within a given time. This is done by the addition of a suitable electrolyte in a suitable quantity.

**Physical character of Settled Slimes.**—While much has been written on the coagulation of suspensions, little or nothing has been done in the study of their physical condition after settling by the various electrolytes. This is a matter of the utmost importance in the cyanide process where decantation is resorted to. We have made some investigations on this subject, and find that the settled slimes vary in bulk, and in the readiness to again become suspended in the solution, with the electrolyte used. The slimes of different ores also vary with the same electrolyte.

The choice of electrolytes capable of being employed in the cyanide process is limited, as they should meet the following conditions:—

1. Be inexpensive.
2. Have little or no detrimental chemical action on the cyanide, nor react with the slimes.
3. Should carry the slimes down to as small a bulk as possible within a reasonable time.
4. Should carry down as little free cyanide as possible.
5. Should not precipitate the dissolved gold or silver.

The quantity of the electrolyte used seems to have great influence on the bulk after settling, and there is a percentage of each substance that has a

maximum value, beyond which its action is detrimental. This maximum is generally a small quantity, but varies with the ore and the electrolyte used. The following experiment illustrates this effect.

Two cylinders were charged with the same mixture and equal quantities of slime in cyanide solution (1 of slime to 3 of solution). To one charge was added 1 grm. of caustic lime and to the other 1 grm. of chalk, equal to 1 lb. per ton dry slime. In four hours the charge containing chalk had settled  $1\frac{1}{2}$  inches, while the charge containing the lime had settled only  $\frac{1}{2}$  inch; thus showing that the chalk was nearly twice as effective as lime. The solution containing lime was, however, perfectly clear, while that containing chalk was slightly cloudy. In twenty-four hours the chalk settled the slime  $5\frac{1}{2}$  inches and the lime only  $3\frac{1}{2}$  inches. In forty-eight hours the chalk settled the slime 6 inches and the lime 4 inches.

Two grams of lime and two grams of chalk were next added to their respective cylinders and well shaken. In eight hours the cylinder containing chalk had subsided  $\frac{1}{2}$  inch and lime  $\frac{1}{4}$  inch. In forty-eight hours that containing chalk subsided  $1\frac{1}{2}$  inches, and that containing lime  $1\frac{1}{2}$  inches; thus showing how detrimental is the action of an excess of the coagulant. On examining the physical character of the settled slimes, it was observed that the slime coagulated with lime was very easily disturbed, and appeared to be somewhat loose and flocculent when compared with that coagulated with chalk. The particles in the latter case had assumed a more granular form, and when disturbed again settled readily, thus being more favourable for separation of the solution by decantation. At the end of a week the cyanide solution containing lime was 0.09 per cent. and quite clear, while in the case of chalk it was only 0.03 per cent. and amber-tinted, showing that the lime was the better protector. The  $\text{CO}_2$  ions, which are introduced by the chalk, are known to facilitate hydrolytic action in the cyanide solution and produce compounds that give it an amber tint.

The cubic contents of the slimes settled in a given time have been observed to be dependent on the valency of the positive ion of the electrolyte, on the quantity of the electrolyte used up to a maximum, and on the temperature. The value of hydrates for this purpose has been found to be generally less than other salts of the same metal. It is noteworthy, from a practical point of view, that within ordinary working limits the cubic contents of settled slimes is independent of the depth of the column, although the time of settling is often said to be inversely as the square of the depth. In practice a moderately deep column is preferred, as it allows a larger amount of clear liquid to be drawn off without disturbing the settled slimes.

The temperature effects have also been observed, and it was found that at about freezing point the settling is very slow, while of course at boiling point it cannot settle at all. Between these two limits there is a point that gives a minimum time of settling and a minimum bulk in a given time, but for practical purposes it is not advisable to exceed  $130^\circ\text{F}$ . It may be taken, in a general way, that the higher the temperature the more readily do slimes



settle. After boiling and allowing to cool, slimes settle rapidly, especially on the addition of a suitable coagulant.

A number of electrolytes were examined by various authorities for their efficiency in settling slimes, of which the following is a resumé.

**Settlement of finely divided Solids suspended in Water.**—It has been known for many years that the capacity of water for holding matter in suspension is more or less destroyed by the addition of certain acids, salts, and alkalies.

In 1868 Wm. Skey \* published the following results of experiments on the precipitation, by neutral salts, of clay suspended in water.

(a) A strong solution of the chloride of sodium, ammonium, calcium, magnesium or barium, or of sulphate of soda, causes an immediate aggregation of the particles, and their complete precipitation shortly afterwards.

(b) "The volume of clayey water clarified by one grain of certain of the above named salts in 24 hours is approximately as follows" † :—

			Proportion.	Per cent.
1 grain common salt	clarifies 5 oza.	= 1 part in 2,187·5	...	0·046
1 " sodium sulphate	" 5 "	= " "	...	0·046
1 " barium chloride	" 10 "	= " 4,375	...	0·023
1 " lime	" 15 "	= " 6,562·5	...	0·014
1 " sulphuric acid	" 50 "	= " 21,875	...	0·005

(c) "The quantity of clayey matter present appears of secondary importance, complete precipitation having nearer relation to the degree of dilution allowed to the salt employed."

This last statement is a very important one to remember in connection with slime treatment.

In 1874 Dr Sterry Hunt ‡ found that the clay suspended in Mississippi water took from 10 to 14 days to settle, but that the addition of sea water, or of salt, magnesium sulphate, alum, or sulphuric acid reduced the time required for clarification to about 12 or 18 hours.

In the same year Wm. Durham, F.R.S.E., published some experimental results,§ and arrived at the following conclusions :—

(d) In solutions of sulphuric acid and sodium chloride the liquid cleared "in the order of the specific gravities of the solutions, so that the densest liquid settled and cleared last." This effect was more decided with acid than with salt.

(e) The suspending power of water was "*gradually increased* by the addition of small quantities of the alkalies or their carbonates, or lime. Thus water having 3 grains of sodium carbonate in it" (in about a pint) "was quite opaque for three days, while water only was seen through in a day and a half." "In sodium carbonate solutions, and most probably in all alkaline solutions,"

\* "Coagulation and Precipitation of Clay by neutral Salts generally," *Chem. News*, xvii. p. 160.

† We have added the last two columns from Skey's figures.

‡ *Chem. News*, xxx. p. 97.

§ *Ibid.*, p. 57.

the greater part of the clay sank to the bottom, and the liquid cleared "in the inverse order of the specific gravities, so that the densest liquid settled and cleared first."

That is to say, with acids, small quantities give the best results, and with alkalis, larger quantities produce quicker settlement. There is, however, an upper limit, beyond which a further increase in the quantity of alkali gives no better or quicker settlement.

Table XLIV. gives some quantitative results of Durham's experiments with suspended clay, to which we have added the last two columns.

### TABLE XLIV.

	Time of clearing. Hours.	Density.	One Part in	Per cent.
Water only	36	1000	...	...
" with 2 drops of $H_2SO_4$	0.5	1000	...	...
" with $H_2SO_4$	1.5	1024	...	2.34
" " " " "	5	1048	...	4.58
" " " " "	10	1093	...	8.51
" " " " "	36	1440	...	30.56
Water with 1 grain $Na_2CO_3$	96	...	8750	0.011
" " 5 " " "	112	...	1750	0.057
" " 9 " " "	98	...	972	0.103
" " 20 " " "	46	...	437.5	0.228
" " 30 " " "	22	...	292	0.342
" " 200 " " "	4	...	43.75	2.235

The jars used in these experiments held about a pint of water. The same general results were obtained also with finely powdered silica, which, however, in all cases settled more rapidly than clay.

In 1878\* Durham published the following additional facts (amongst others) relating to the same subject.

(7) "Clay in strong  $H_2SO_4$  was suspended nearly as long as in water, but on mixing the liquids the precipitation of the clay was greatly accelerated." Both Skey and Durham conclude that suspension of clay in water is due to a weak chemical affinity between them, which is overcome by the stronger affinity of water for the acids, salts, and alkalies. These views are, however, gradually giving place to the electro-chemical explanation referred to on page 204.

From a most interesting paper by M. J. Thoulet,<sup>†</sup> published in 1891, we extract the following results of a few of his many careful experiments, which were made with very fine kaolin in a graduated glass tube, placed in a d'Arsonval stove to ensure constant temperature during each settlement. The depth of clear liquid measured from the top was read off from time to time, as a measure of the subsidence.

\* *Chem. News*, xxxvii, p. 47.

† *Annales des Mines*, vol. xix. p. 1.

(g) His first experiments show that in vessels of considerable size and depth, the rate of subsidence during the first part of the descent is almost constant, and that the retardation that becomes measurable later is a function of the depth, measured from the top of the still turbid portion of the liquid to the bottom of the vessel. This agrees with the fact that in cyanide practice the settlement of slime in a vat is much slower when nearly complete, than it is at the beginning of the operation.

(h) Table XLV. shows that the rate of settlement increases with the temperature.

TABLE XLV.

Temperature degrees C.	Observed Fall in 80 hours. In millimetres.	Fall averaged from plotted Curve.	Difference.
52	24.25	23.50	-0.75
55.5	25.00	26.25	+1.25
56.5	27.00	27.00	0.00
62	33.00	31.75	-1.25
66.5	34.00	35.25	+1.25

The plotted curve referred to is given in the original paper, and is a straight line which cuts the horizontal axis at 23°, and thereby suggests that possibly at that temperature the clay would remain indefinitely in suspension.

(i) In experiments with HCl as a precipitant, Thoulet found that the rate of fall increased nearly in proportion to the quantity of acid used, a result which appears to be directly contrary to those of Durham with H<sub>2</sub>SO<sub>4</sub>. Thoulet does not give the exact proportion of acid, but it appears to have been 5 c.c., 10 c.c., and 15 c.c. pure HCl in three experiments respectively to less than 70 c.c. of turbid water in each case.

(k) Experiments with variations in the amount of kaolin showed that an increase of suspended matter causes a decrease in the rate of subsidence up to a certain limiting quantity, not stated, beyond which the rate of subsidence is independent of the quantity of solids in suspension.

(l) Experiments with increasing quantities of sea water added to the clayey water showed an increased rate of subsidence up to 10 per cent. of sea water. This mixture had a specific gravity of 1.00253, and produced an immediate clarification equal to that effected by sea water alone.

Alum has been in use for some time as a coagulant, to settle gold slimes in the clean-up from the zinc boxes, and it has been proposed for settling battery slimes.

Soap as a coagulant was proposed by Sulman and Teed, but has not been adopted, principally on account of expense.

It is clear from the foregoing that different substances have different coagulating powers, and the problem at once presents itself, to find what is the relative coagulating power of the different electrolytes.

We have investigated the effect of a number of substances, which we give in Table XLVI. The figures represent the relative weights required to produce the same coagulating effect, i.e. the relative efficiency of the electrolyte.

TABLE XLVI.

Substance.	Relative efficiency.
Aluminium sulphate, . . .	100
Alum (potash), . . . . .	143
Ferric iron, . . . . .	223
Alum (ammonium), . . . . .	252
„ (am. chrom. iron), . . . . .	295
Lime, . . . . .	654
Magnesia, . . . . .	748
Alum (pot. chrom.), . . . . .	858
Calcium chloride, . . . . .	1,095
„ carbonate, . . . . .	1,215
„ sulphate, . . . . .	2,870
Magnesium sulphate, . . . . .	3,460
Sodium chloride, . . . . .	45,900
„ sulphate, . . . . .	61,700

When the cost of each of these substances is known, it will be an easy matter to determine which is the most economical to employ.

At the present time, lime may be said to hold the field at the chief gold-mining centres, as the cheapest and most efficient settling agent. It is usually added to the pulp outflowing from the sand settling tanks, in the launder that leads to a very large spitzkasten, which is provided for the separation of the slime. With the use of slime, if this spitzkasten is properly proportioned, practically the whole of the lime is contained in the underflow in about 12 to 15 times its own weight of water, while the remaining water overflowing from the top of the spitzkasten is completely clarified, and may be at once returned to the battery. For sizes of slime spitzkasten and their construction see Chapter XLI.

**Secondary uses of Lime as a Coagulant.**—The main objects of the use of lime as a coagulant are to obtain the slimes quickly in their treatment vats, to avoid the partial oxidation which results from collection in large dams, to save the cost of re-handling, and to clarify the water for immediate use again. But in addition to these, the following indirect advantages resulting from its use have been pointed out by J. R. Williams.\*

(1) A larger percentage of gold is caught by amalgamation, because the lime in the battery water, by coagulating the slimes, causes these extremely fine particles of ore to come into contact with the plates.

(2) Less time is required in treatment of the sands, because they are com-

\* *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii. p. 57.

pletely neutralised by the lime water with which they are crushed and transported to their vats.

(3) An actual saving of water, which is otherwise lost by evaporation in the slime dams.

To these we may add in many cases—

(4) A saving of the cost otherwise incurred for pumping water back to the battery from the main water dam, which is usually of necessity much lower than the slime plant.

The first of the above statements has been seriously questioned, but it is supported by the following figures :—

TABLE XLVII.

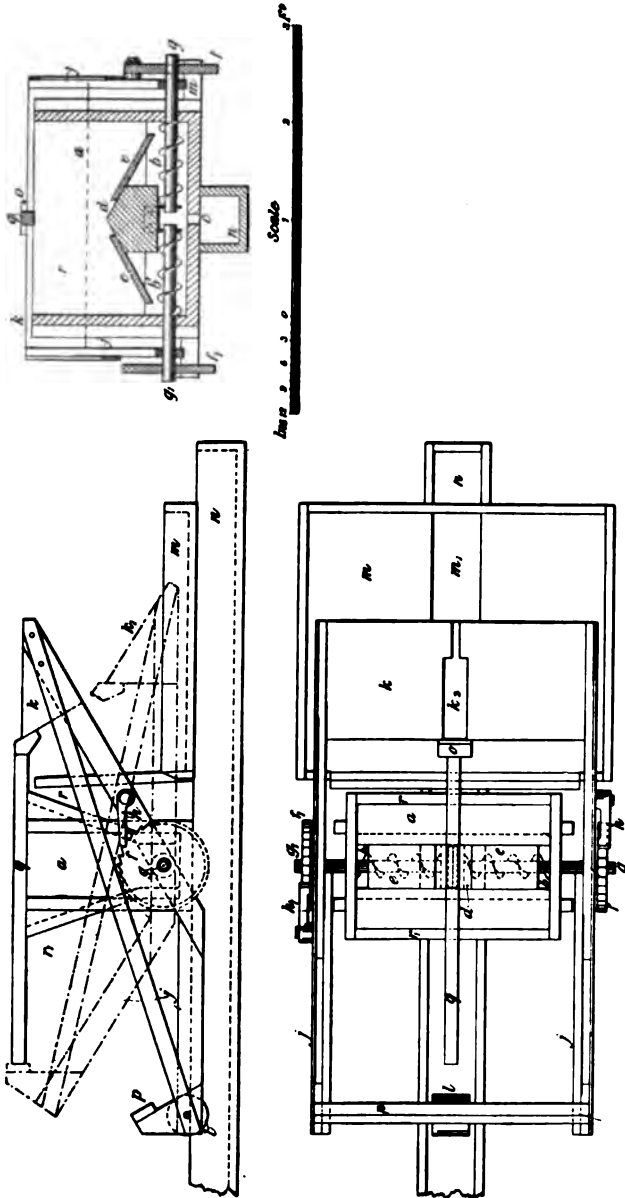
Name of Mine.	Percentage of total Gold caught by Amalgamation.		Value of Slimes in dwts.	
	Without Lime.	With Lime.	Without Lime.	With Lime.
Geldenhuis Deep, . . .	49·68	54·58	4·21	2·98
Treasury, . . . . .	55·60	61·50	4·01	2·55
Crown Reef, . . . . .	50·01	57·81	6·16	4·50
Bonanza, . . . . .	55·71	59·81	8·00	5·75

The reduction in the value of slimes resulting from the use of lime is of great importance when the building of a slimes plant is contemplated, for it is evident that a knowledge of the assay value of the slimes, obtained without lime, is not a safe basis for the calculation of the economic results of such a proposed plant. It is therefore necessary to ascertain by the experimental use of lime in the mill water, in cases where it is not already employed, whether the slimes under the new conditions will be rich enough to pay for treatment.

The use of lime is also said to increase the quantity of sand caught in the tailings vats, by an amount variously estimated by different authorities, at from 2 to 5 per cent., and also to render the sands more leachable.

The quantity of lime found necessary for the complete clarification of the battery water when the process first starts may be very considerable, and the regular consumption after a circulation of neutralised water has been established will vary with the acidity of the ore and other factors, so that no exact quantities can be given. On the Rand, however, the quantity of lime used per ton of water passing the spitzkasten is generally between 0·1 and 0·2 pounds. As the quantity of slime also varies relatively to the water, the consumption of lime in current work may vary at different places, say, from 3 to 9 lbs. per ton of dry slime treated.

**Feeding the Lime.**—As the lime, for economical reasons, should always be ground before use, a small ball mill may be arranged, not only to do the



FIGS. 46, 47, 48. —Tavener's Automatic Lime Feeder, as used at the Bonanza Mine, Johannesburg.

grinding, but also to deliver its product into the launder conveying the slime and water. By regulating the speed of the mill, the rate of delivery can be easily adjusted. But the mill, of course, requires regular and frequent feeding by hand.

An automatic apparatus for lime feeding, which only requires filling up at long intervals, has been designed by P. S. Tavener, and is illustrated in figs. 46, 47, 48. This machine has a wedge-shaped hopper *a* which is filled with powdered lime. The hopper has inside it two helical threads *b b*<sub>1</sub> at the bottom (on separate shafts *g g*<sub>1</sub>), which push the lime from each side towards a central discharge hole *c*. This hole and the inner ends of the threads are covered by the block *d* and the inclined boards *e e*, so that by adjusting the position of these boards on the block the supply of lime to the threads *b b*<sub>1</sub> can be roughly regulated. Ratchet wheels *f f*<sub>1</sub> are fixed to the shafts *g g*<sub>1</sub> and are actuated by pawls *h h*<sub>1</sub> attached to a rocking frame *j*, which is loosely centred on the projecting ends of the shafts *g g*<sub>1</sub>. A water box *k* is attached to one end of the rocking frame, and a counterweight *l* at the other end. The action of the apparatus is as follows, assuming that the rocking frame is in the position shown by full lines in fig. 46.

A small stream of water falls into the opening *k*<sub>2</sub> of the water box, which, when sufficiently full, falls down on to the tray *m* as shown at *k*<sub>1</sub>, thus moving the rocking frame into the position shown by dotted lines. During this motion the pawl *h*<sub>1</sub> pushes round ratchet wheel *f*<sub>1</sub>, the shaft *g*<sub>1</sub>, and thread *b*<sub>1</sub>. The water in the box *k* escapes through opening *k*<sub>2</sub> into tray *m*, and thence through hole *m*<sub>1</sub> into the lime water launder *n*. When the box *k* is empty the counterweight brings back the rocking frame to its first position, and during this movement wheel *f*, shaft *g*, and thread *b* are actuated. Thus at each movement of the rocking frame a small quantity of lime is fed towards the discharge opening *c* and falls into the launder *n*, whence it is washed into the slime launder at any convenient point by the water which is used to work the apparatus.

The block *o* on the water box and the bar *p* at the other end of the frame alternately strike the rod *q* at the end of each movement, and this rod is attached to the two inclined sides *r r*<sub>1</sub> of the lime hopper, which are hinged so as to be capable of a limited motion. The movement of these hinged sides each time the rod is struck prevents the lime from packing, and shakes it down to the openings at the lower edges of the inclined boards *e e* over the outer ends of the threads *b b*<sub>1</sub>. Thus the action of the apparatus is entirely automatic, and it has proved quite successful in practice. The feed can be further regulated by altering the supply of water to box *k*, or by altering the size of its outlet at the narrow end.

## CHAPTER XXX.

### DISSOLVING THE GOLD AND SILVER IN SLIMES.

As only the finest gold and silver particles originally present in the ore can escape from the battery plates and sand-collecting apparatus, it would seem at first sight that the problem of dissolving these metals from slime was a very simple one. But there are two important points of difficulty that were discovered only when treatment had been carried out on a large scale. These difficulties were ably discussed by Chas. Butters in his valedictory address to the Chemical and Metallurgical Society of South Africa\* when describing the development of the slime process.

**The Effect of Viscosity of Pulp.**—The first trouble is chiefly a mechanical one, of which Butters says :—"One important point in the solution of the gold was not really understood until we had been working on a large scale for a long time ; that is, the concentration of the slimes in the liquid, or in other words, the thickness of the pulp. A ton of liquid which is agitated with half a ton of slimes forms a smooth viscous pulp, in which, if you suspend a can of cyanide (which dissolves readily in water), the lumps of cyanide remain for several hours undissolved. In other words, the thick pulp will not readily dissolve even so soluble a substance as cyanide, while if you thin the pulp to four or five tons of liquid to one of slime, soluble substances will dissolve very rapidly." The above analogy to the solubility of gold is perhaps not quite an accurate one, but it forcibly presents to the practical mind the effect of thick pulp in diminishing dissolving power of a solvent.

We have made some determinations on the solubility of gold in a KCy solution containing slime in different ratios, to show the effect due to viscosity. These results are given in Table XLVIII. and are plotted in fig. 49.

TABLE XLVIII.

Ratio of Solution to 1 of Slime by weight.	Rate of Dissolution of the Gold.
KCy Sol. alone.	100
6	58·6
5	56
4	52·4
3	47·3
2·5	44·3
2	40
1·5	34·2
1	20

\* Society's *Jour.*, vol. ii. p. 5, Feb. 1898.



The strength of the solution in each case was 0.11 per cent. KCy. The slime was free from organic matter and soluble sulphides, so that the loss in solubility of the gold with increase in thickness of pulp may be considered to be chiefly due to increased viscosity.

The reason of increased viscosity reducing the solubility would appear to be largely due to a retarding influence on the movements of the K and Cy ions. Agitation by stirrers diminishes this influence but little when the pulp is thick, as the mass tends to move as a solid whole rather than to diffuse.

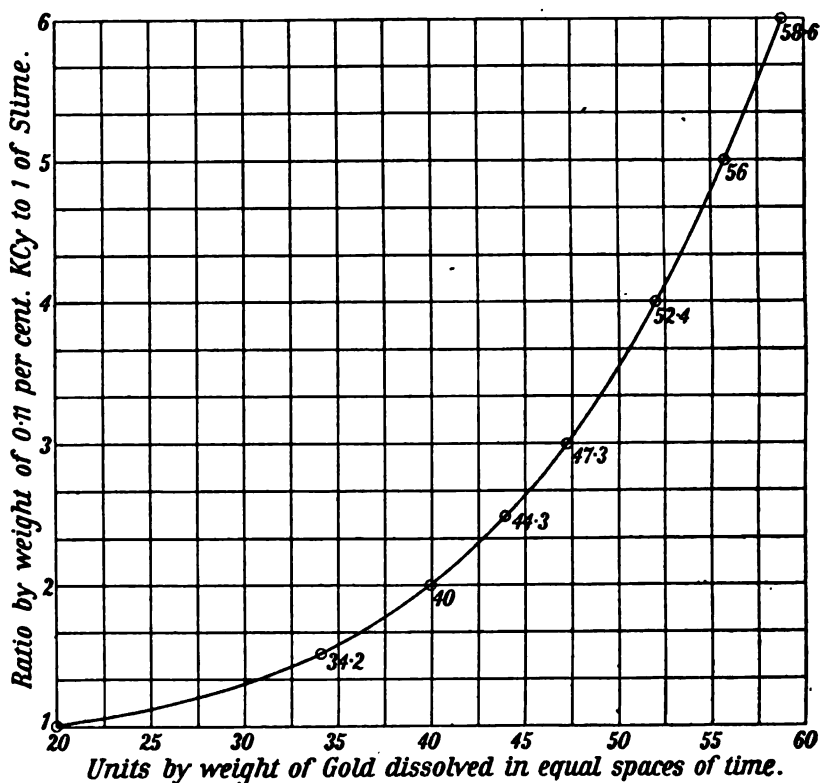


FIG. 49.—The Effect of Viscosity of Pulp on Dissolution of the Gold.

Another source of loss in dissolving power is that the action of the fine solid particles is to dissociate the KCy molecules and adsorb the potassium, leaving the cyanogen free to combine with water. Now, cyanogen is not a solvent for gold, and when it combines with water it forms HCy and HCyO. The former has but a slight action on gold, and the latter less. When excess of alkali is present we get KCy and KCyO, and this accounts to some extent for the fact that, when dissolving gold from slimes, a pure cyanide solution without the addition of alkali or an alkali earth always shows a poorer extraction.

**The Uses of Oxygen in Thick Pulp.**—The second difficulty is chiefly a chemical one, due to the presence of substances capable of combining with absorbed oxygen.

In fresh slimes we have only to deal with the usual products of partly decomposed minerals, but slimes being in a very minute state of subdivision, are much more acted on than the same weight of the same substance in a coarser condition, as found in sands. On the other hand, old slimes which have been lying for a considerable period, sometimes many years, in contact with surface water, and often overgrown with vegetation, contain quantities of organic matter that act as reducing agents, and combine with the absorbed oxygen in the solution. Further, it has been pointed out by Caldercott \* that slimes lying in dams containing pyrites undergo a partial decomposition, with formation of ferrous sulphide ( $\text{FeS}$ ), sulphurous acid ( $\text{SO}_2$ ), and sulphuretted hydrogen ( $\text{H}_2\text{S}$ ), all of which act as reducers and rob the solution of absorbed oxygen.

Butters † says six to twelve hours' agitation is sufficient to dissolve the gold in fresh slimes, whereas forty-eight hours with accumulated slimes failed to do so. This is undoubtedly owing to want of oxygen, which becomes used up in oxidising reducing agents. Therefore, in treating old slimes, an effective supply of oxygen by aeration or other means is an imperative necessity, and in the treatment of all slimes its value can hardly be overrated.

When oxygen is introduced into the pulp, its use is first to oxidise any reducing matter, such as soluble sulphides and organic matter, and as this is effected, the dissolution of the gold and silver takes place. When all the reducers are completely oxidised, the time required to dissolve the gold in true slime is very short, often less than an hour, provided oxygen is supplied as wanted. Owing to this, strong oxidisers may beneficially be added to the pulp, as the gold dissolves completely before they have lost the whole of their oxidising power, whereas if strong oxidisers are applied to coarse particles, the time that they are effective is only a fraction of the whole time required to dissolve the gold.

**Velocity of the Reaction.**—When an energetic oxidiser is added to a dense solution, the velocity of the reaction is slower than when added to a dilute solution. Thus, when a strong oxidiser is added to an ordinary clear cyanide solution, the velocity of the reaction may be so rapid that its oxidising effect is many times as fast as demanded by the dissolving metal, and therefore waste goes on, but when the same oxidiser is added to slime pulp, the velocity of the reaction is enormously reduced, so that we may then have the oxidising effect produced at a rate which approximates to that required to dissolve the metal.

**The Oxidation or Aeration of the Pulp.**—Some operators have found it an advantage, when dealing with accumulated slimes, to first add to the pulp a considerable quantity of a strong oxidiser, and then run in the cyanide

\* *S. African Mining Jour.*, Oct. 23, 1897.

† *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. p. 5, 1898.

solution. Others have done the oxidising by blowing air through the pulp. Various mechanical means for the aeration of the pulp have been used, and we shall now give a short account of these.

One of the first methods was to force air into the pulp by means of a compressor, through a perforated pipe fixed at the bottom of the vat. This was done before and after the addition of the cyanide to the pulp, and has been worked on a large scale, but was superseded by a cheaper means of aeration introduced by H. T. Durant,\* namely, the introduction of air into the centrifugal pumps, used to circulate and transfer the slime pulp. He placed a valve in the suction pipe of the pump, and between this valve and the pump inlet he fixed a small air inlet valve. By partially closing the main valve the supply of pulp is restricted to less than the full capacity of the pump, and consequently the latter draws in a small but regular supply of air. This method should, of course, only be applied to a pump fixed below the level of the pulp supply, otherwise the air would interfere with the action of the pump.

At the present time the Rand Central Ore Reduction Company have given up the above method, and force a small quantity of air at low pressure into the pulp delivery pipe immediately above the centrifugal pump, in order to avoid the loss of efficiency of the latter caused by admitting air to the suction pipe. But none of the above methods has proved entirely satisfactory.

\* *Jour. Chem. and Met. Soc. S. Africa*, vol. ii. p. 59.



At the first decantation, the fraction of liquor drawn off, and consequently the fraction of gold extracted, is

$$\frac{a}{a+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

and the gold remaining in the vat is

$$1 - \frac{a}{a+m} = \frac{m}{a+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

At the second decantation we shall extract of the gold *then present* the fraction

$$\frac{b}{b+m};$$

therefore the fraction of total gold which is obtained by this second decantation is

$$\frac{m}{a+m} \times \frac{b}{b+m} = \frac{bm}{(a+m)(b+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

and the fraction of total gold which remains in the vat will be

$$\frac{m}{a+m} - \frac{bm}{(a+m)(b+m)} = \frac{m^2}{(a+m)(b+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

Proceeding in the same way, the fraction extracted at the third decantation is

$$\frac{cm^2}{(a+m)(b+m)(c+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (6)$$

and the fraction remaining in the slime is

$$\frac{m^3}{(a+m)(b+m)(c+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (7)$$

Finally, if  $l$  is the quantity of liquor drawn off at the last operation, and there are  $n$  decantations altogether, we can put down general expressions for both quantities thus:—

The fraction extracted by the  $n$ th decantation is

$$\frac{lm^{n-1}}{(a+m)(b+m)(c+m) \dots (l+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (8)$$

and fraction of total dissolved gold left in the residues is

$$\frac{m^n}{(a+m)(b+m)(c+m) \dots (l+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (9)$$

From the above expressions the total extraction can be calculated in two ways, either by adding together the separate extractions, namely (2), (4), (6), . . . . . (8), or by subtracting the residual gold in (9) from unity. Adopting the latter method, we get for the total extraction after the  $n$ th decantation, which we may call  $E$ , the following expression:—

$$E = \frac{(a+m)(b+m)(c+m) \dots (l+m) - m^n}{(a+m)(b+m)(c+m) \dots (l+m)} \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (10)$$

Now, it is evident that as  $a$ ,  $b$ ,  $c$ , etc. are positive quantities, the expression  $(a+m)(b+m)(c+m) \dots (l+m)$  is always greater than  $m^n$ , and that therefore the maximum value of  $E$  is coincident with the maximum value of

that expression. But the sum of these factors is equal to  $s - m + mn$ , which in any given case is a constant quantity. Consequently  $(a + m)(b + m)(c + m) \dots (l + m)$  has its maximum value when all its factors are equal, and we arrive at one useful practical result, viz.—*With any quantity of solution and any number of decantations, the maximum extraction is obtained when equal quantities of solution are decanted at each operation.*

With equal decantations let  $q$  = the quantity of liquor drawn off at each operation, then

$$q = \frac{s - m}{n}$$

and therefore in this case equation (10) becomes

$$E = \frac{(q + m)^n - m^n}{(q + m)^n} = \frac{\left(\frac{s - m}{n} + m\right)^n - m^n}{\left(\frac{s - m}{n} + m\right)^n} \quad \dots \quad (11)$$

From this the total theoretical extraction can be calculated for any given values of  $q$ ,  $m$ , and  $n$ , or of  $s$ ,  $m$ , and  $n$ .

And again, because  $\frac{s - m}{n} + m$  is necessarily greater than  $m$ , it follows that

with a fixed value of  $s$  and increasing values of  $n$ , the value of  $E$  will continually increase, always approaching, but never reaching, unity. Therefore *a given total quantity of liquor will give the highest extraction when divided into the largest practicable number of separate washes.*

By careful settlement and decantation in the laboratory, using sufficient lime, the moisture remaining can be reduced to about 30 per cent. of the pulp remaining in the vat, which gives the value,  $m = 0.43$ ; but in practice it is very generally found that after decantation the remaining mass contains 50 per cent. of water. In this case  $m = 1$ , and from equation (11) we get

$$E = \frac{(q + 1)^n - 1}{(q + 1)^n} = \frac{\left(\frac{s + (n - 1)}{n}\right)^n - 1}{\left(\frac{s + (n - 1)}{n}\right)^n} \quad \dots \quad (12)$$

As it is usual to express results in percentages, let  $P$  represent the total extraction per cent. of the dissolved gold, then—

$$P = 100E \quad \dots \quad (13)$$

From the above equations, Tables XLIX. and L. have been calculated for the values  $m = 1$  and  $m = 0.75$  respectively, in order to exhibit in a clear and concise manner the effect of varying the number and quantity of the solutions, under all the conditions which are likely to occur when working with natural settlement.

The first fact exhibited by Table XLIX. is that, although with any given quantity of liquor, as already stated, the extraction can be always increased with increased subdivision of the liquor, yet when small quantities are used, the rate of increase is so slow that a good extraction cannot be obtained with

any reasonable number of decantations. For instance, with 2 tons of liquor per ton of dry slime, divided into 5 parts, the total extraction after the fifth decantation is only 60 per cent., and to reach a theoretical extraction of 95 per

TABLE XLIX.—*Showing the Total Percentage of the Dissolved Gold which is extracted from Slime by n equal decantations, when the residual pulp contains 50 per cent. of moisture, i.e. when  $m=1$ .*

$s$  = total quantity of liquor used, including original moisture.

Value of $s$ .	Number of Decantations.				
	1	2	3	4	5
2	50	55.56	57.81	59.02	60
3	66.67	75	78.4	80.25	81.41
4	75	84	87.5	89.34	90.48
5	80	88.89	92.13	93.75	94.71
6	88.33	91.84	94.73	96.10	96.88
7	85.71	93.75	96.30	97.44	98.06
8	87.5	95.06	97.30	98.25	98.75
9	88.89	96	97.97	98.77	99.16
10	90	96.69	98.44	99.10	99.42
11	90.91	97.22	98.77	99.38	...
12	91.67	97.63	99.02	...	...

cent. it would be necessary to divide the solution into 200 parts, with 200 decantations, which of course is in practice absurd.

Here, then, we have another reason why comparatively large volumes of solution must be used when treating slime by natural settlement. The table

TABLE L.—*Showing Percentage Extraction when  $m=0.75$ , or about 43 per cent. of moisture in residue after each decantation.*

Value of $s$ .	Number of Decantations.				
	1	2	3	4	5
5	85	93.19	95.85	97.07	97.74
6	87.5	95.06	97.3	98.25	98.74
7	89.29	96.25	98.14	98.89	99.26
8	90.63	97.06	98.67	99.27	99.76
9	91.67	97.63	98.97	99.49	...

shows, however, that with 6 tons of liquor per ton of slime and four decantations, or with 10 tons divided into two parts, an extraction of over 96 per cent. can be obtained. And these figures may safely be said to represent the extreme practical limits of the method of decantation after natural settlement.

To decide exactly what values will give the most profitable result in any particular case, it is necessary to know the cost of each extra decantation, including, of course, the stirring and settlement, and also to know the cost of precipitating the gold from one ton of solution.

For example, suppose that the cost of one extra decantation is equal to 6d. per ton of dry slime, and that the cost of precipitating from each ton of solution is 3d. Then the following figures show the relative costs of several variations within the practical limits of working. It is to be remembered that one long agitation for dissolving the gold followed by one decantation is necessary in any case.

(a) 6 tons of solution in 4 parts, giving 96·10 per cent. extraction :

Cost of precipitating 6 tons at 3d.,	. . .	1s. 6d.
„ three extra decantations at 6d.,	. . .	1s. 6d.
Total,	. . .	3s.

(b) 7 tons of solution in 3 parts, giving 96·30 per cent. extraction :

Precipitating 7 tons of liquor at 3d.,	. . .	1s. 9d.
Two extra decantations at 6d.,	. . .	1s.
Total,	. . .	2s. 9d.

(c) 9 tons of solution in 2 parts, giving 96 per cent. extraction.

Precipitating 9 tons at 3d.,	. . .	2s. 3d.
One extra decantation at 6d.,	. . .	6d.
Total,	. . .	2s. 9d.

At these prices *b* and *c* are equal as to cost, but as *b* gives a better extraction and takes less time it is to be preferred. It must be understood, of course, that the prices taken must include interest and depreciation of plant as well as labour and materials. To show how necessary it is to ascertain these prices with accuracy, we may, as another example, take the cost of precipitation at 5d. per ton of liquor, and the cost of stirring, etc. at 4d. Then, for the same three cases, the totals are—

(a) 3s. 6d.                      (b) 3s. 7d.                      (c) 4s. 1d.

Therefore, at these prices, method (a) is preferable unless the material is so rich that the extra 0·2 per cent. recovered by method (b) turns the scale in its favour.

A comparison between Tables XLIX. and L. shows the advantage gained by a reduction in the amount of water retained by the slimes after settlement and decantation ; in the latter table this quantity is equal to 0·75 of the dry weight of the slime. That is to say, the residual pulp only contains about 43 per cent. of moisture after each operation, instead of 50 per cent. as in Table XLIX. Referring again to the three sets of conditions (*a*, *b*, and *c*) selected above, the increases in the extraction are 2·15, 1·84, and 1·63 per cent. respectively. The gain, therefore, is not apparently very great, and is



relatively smaller with the larger quantities of liquor, but it is sufficient to encourage efforts to reduce the quantity of residual moisture, especially in the case of rich slimes.

Although the preceding results are useful as they stand for comparing the respective advantages of different methods of working, yet it must not be forgotten, that they are only strictly accurate when the liquor which is used for washing contains no residual gold from previous operations. But in practice, of course, the solutions generally contain some gold, and we shall therefore briefly examine the loss thereby incurred, on the assumption that all the liquor used in dissolving and washing has the same assay value when applied to the slimes.

If the slime is dry when put into the vats, this loss is *independent* of the number of decantations and of the total quantity of liquor, and depends only upon the quantity of solution  $m$  which is finally left in the slime when the treatment is completed. Thus if  $m=1$ , then for every grain of gold per ton in the wash liquor, one grain will be lost per ton of slime treated.

When treating wet slime, the loss is always less, because of the original moisture, a part of which remains to dilute the final wash. The larger the number of decantations, the more nearly will the loss approach to its greatest limit, which is identical with the amount lost in treating dry slime.

The following equations will enable the amount of this loss to be readily calculated for any values of  $m$  and  $q$ , and for one, two, or three successive operations, which will include all practical cases. The following equations relate *only* to the gold in the solutions before use.

Let  $x$  = assay value of solution added to the slime.

$q$  = as before, the quantity decanted each time.

$m$  = as before, the moisture remaining each time.

Then quantity of gold added to slime with each wash liquor =  $qx$ , and of this we have  $x$  returned at first decantation

$$\frac{qx}{1} \times \frac{q}{q+m} = \frac{q^2x}{q+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad (14)$$

And left in slime after first decantation

$$qx - \frac{q^2x}{q+m} = \frac{mqx}{q+m} \quad . \quad . \quad . \quad . \quad . \quad . \quad (15)$$

Returned at second decantation

$$\frac{q^2x(q+2m)}{(q+m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (16)$$

Left in slime after second decantation

$$\frac{mqx(q+2m)}{(q+m)^2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (17)$$

Returned at third decantation

$$\frac{qx^2(q^2+3mq+3m^2)}{(q+m)^3} \quad . \quad . \quad . \quad . \quad . \quad . \quad (18)$$

Left in slime after third decantation

$$\frac{mqx(q^2 + 3mq + 3m^2)}{(q + m)^2} \quad . \quad . \quad . \quad . \quad . \quad (19)$$

One example may be given :—Assume that 3 tons of liquor assaying  $1\frac{1}{2}$  grains are added and decanted twice, then the loss of gold from this source per ton of slime when  $m=1$  is by (17)—

$$\frac{1 \times 3 \times 1 \cdot 5(3+2)}{4 \times 4} = 1 \cdot 4 \text{ grains.}$$

## CHAPTER XXXII.

### TREATMENT BY AGITATION AND NATURAL SETTLEMENT ON THE RAND.

THIS process is the one that has, up to the present time, been most largely practised, especially on the Rand goldfields, where it was first introduced in 1894. It is the application in a large way of the method of washing precipitates which is used on a small scale in connection with analytical work in the laboratory. The practical development of the process to a commercially successful stage was chiefly due to the work of Chas. Butters and J. R. Williams, the former of whom used revolving stirrers for the purposes of agitation, while the latter relied chiefly upon circulation by centrifugal pumps.

In dealing with accumulated slimes from dams, they are usually first delivered to a pulping vat partly filled with solution. This vat has a stirrer, consisting of two or four arms attached to a vertical shaft, which is driven by bevel gearing from a horizontal shaft passing over the vat or vats. When thoroughly mixed, the pulp is pumped into one of the treatment vats, and there kept in motion until the dissolution of the gold is sufficiently complete. In the old Robinson plant, the slimes from the dams were delivered into trommels fed with water to take out any coarse sand mixed with them, and afterwards passed through two spitzkasten. The first one eliminated the fine sand, while in the second the slimes themselves were settled and sent to the treatment vats.

When the current product of a mill is delivered to the slimes plant from the underflow of a large spitzkasten, it is also generally sent directly into one of the treatment vats through a shoot which delivers the pulp a few feet below the top edge of the vat. It is found that a vat 24 feet in diameter will settle the slime so completely, when receiving the spitzkasten underflow at the rate of 25 cubic feet per minute, that by the time the vat fills and the water overflows this is completely clarified. When sufficient slime has been collected, the slime water is diverted to another tank. The water is decanted off by means of a hinged pipe whose upper end is supported by a float, so that it sinks with the water, and always receives its supply from the top of the liquid.

This method of decantation is better than the use of a vertical row of plugged holes in the side of the vat, which are successively opened as the level of the water is lowered. When the decantation is complete, the necessary quantity of cyanide solution is added to the settled slime, the stirrer is set in motion, and agitation continued until the gold is dissolved.

The remaining operations are the same whether old or new slimes are treated. Before dealing with the latest practice in the use of this method of treatment, it is advisable to briefly summarise some interesting work carried out by Chas. Butters, in the direction of what may be termed "assisted natural settlement."

It was natural that at first he tried filtration as an aid to settlement and decantation, in eliminating the gold-bearing solutions from the slime. The following work was done at the Robinson slime plant.\* A charge of 10 tons of slime, after the gold had been dissolved, was turned into a second agitating vat, 24 feet in diameter, fitted with paddles which could be raised or lowered, and which revolved at about 16 revs. per minute. After each agitation, the paddles were raised and the slimes allowed to settle. Then the liquor was simultaneously decanted from the top and drawn off through the filter at the bottom by a vacuum pump. By this means the moisture is said to have been reduced to 28 per cent. of the residual mass, which remained as a tough leathery layer about 6 to 8 inches in thickness, and so coherent that it could be rolled up on the filter. After adding more solution this combined filtration and decantation was repeated. As the quantities of solution are not stated in the Journal cited, we cannot calculate the percentage of extraction, but although this was said to be satisfactory, the process, as carried out at the Robinson plant, was too expensive for the grade of material treated there, on account of "the cost of keeping the suction pumps in order, the power and attention they required," and the excessive "wear and tear on the surface of the filter." If the objections above quoted are the only ones that caused the economical failure of this method of treatment, it is probable that the last word has not been said upon the matter, because the vacuum pump difficulty could be overcome by adopting the New Zealand practice of working with a separate vacuum chamber, as described in Chapter XXVII.

Of course, the cost of power for producing the vacuum and the cost of filter cloths still remain to the debit of the method; and in order to see what advantage may be credited against this, the following table (LI.) has been worked out for the case of residual pulp containing only 28 per cent. of moisture, that is, when  $m = 0.4$ .

A comparison of these figures with Table XLIX. shows that two decantations assisted by filtration give, in all cases, a better extraction than three operations with natural settlement only. Also, it is to be noted that with any given total quantity of solution up to seven times the weight of the slime, *one* operation gives a better result than two decantations with natural settlement. With solution equal to eight times the weight of the slime, one operation gives 95 per cent. extraction of the dissolved gold; and as in this case only one agitation is required, this could be effected in a vat without a filter, and when the gold is sufficiently dissolved, the pulp could be transferred to a filter vat for settlement, decantation, and filtration. By this means the wear and tear of filter cloths would be reduced to a minimum.

\* *Jour. Chem. and Met. Soc. of S. Africa*, vol. i. p. 6,—Valedictory Presidential Address.

However, in consequence of the failure of the first attempt in this direction, the use of auxiliary filtration has for the present been abandoned in favour of natural settlement and decantation. The result is, that when the treatment in the agitating vats is finished, and the last settlement carried as far as possible, each ton of slime still retains one ton of solution, which not only contains gold, but also a small amount of cyanide. This latter is, however, so small that it need not be considered. In order to reduce the amount of residual moisture and thus increase the gold extraction, Chas. Butters introduced the use of very large and deep tanks for final settlement before dumping the residues. These tanks have been in some cases 50 feet in diameter and 16 feet deep. The treated pulp from the agitating vats, when the last decantation is nearly completed, is delivered into these large vats by a pipe whose outlet is a few feet below the top of the vat, so that, as the solid slime

TABLE LI.—*Extractions obtained by combined Decantation and Filtration, calculated from preceding formulæ for value of  $m = 0.4$ .*

Total Solution per ton of Dry Slime.	Number of Decantations.	
	1	2
4	90	96.89
5	92	97.81
6	93.33	98.44
7	94.29	98.83
8	95	99.09

settles down, the water rises and a clear water overflow is obtained. The slime at the bottom is compressed by the weight of the slime above, so that when the tank is filled with slime to a depth of about 10 feet, the average moisture is about 40 instead of 50 per cent., after the remaining clear water on top has been decanted off. Water pumped through a 3-inch nozzle at a pressure of 200 lbs. per square inch is sufficient to discharge 500 tons of slime from such vats in three hours.

The following example will give some idea of the actual cash value of this extra settlement, and will, at the same time, further illustrate the use of the tables and formulæ previously given.

Say one ton of slime worth 4 dwts. is treated with eight tons of liquor\* at 3 grains per ton, divided into two washes. Of the 96 grains of gold present in the slime, we may assume that 90 grains will be dissolved. The extraction by natural settlement in the shallow treatment vats will, by Table XLIX., be 95 per cent. = 85.5 grains, thus leaving behind 4.5 grains of the gold dissolved from the slime. From equation (17) we find the residual gold due to the solution used is 2.85 grains; therefore each ton of solution saved is worth

\* i.e., 1 ton of water originally in slime + 7 tons of added solution.

7.35 grains, or 14.7 pence. By the use of the deep tanks one-third of a ton of moisture is saved per ton of slime = 4.9 pence. The cost of precipitation of this extra solution will be about 1d., and we may put the cost of sluicing at 0.9d. per ton of dry slime, so that the account stands thus :—

Extra gold saved per ton of slime, . . . . .	4.9 pence
Extra cost incurred, . . . . .	1.9 „
<hr/>	
Balance in favour of use of deep tanks for additional final settlement, . . . . .	3 pence

There is another advantage arising from the use of these tanks which must be referred to briefly. It is well known that when settling slimes from comparatively large volumes of solution, the settlement is very rapid at first, but becomes extremely slow towards the end as the pulp thickens. Where extra tanks are not provided, each charge must remain in its treatment vat until the natural settlement is completed down to 50 per cent. of moisture. But where the extra tanks are in use, a part of the settlement and decantation may be performed in the treatment vat, and the charge then transferred to the large tank, in which several charges will be simultaneously settling, whereby time is saved, and an equal number of treatment tanks are set free to receive fresh charges. Of course, the total cubic capacity required in the plant for settlement purposes must be the same in either case, but this capacity is more cheaply provided in the form of large plain tanks than in the form of treatment vats, with stirrers and gearing, which are idle while settlement is going on.

This principle can doubtless be carried further by operating with still deeper vats, probably of small diameter, provided with means of withdrawing, at intervals or continuously, the lowest and therefore driest stratum of settled slime. But this form of removal might easily cost more than the additional gold recovered as compared with the present practice.

One of the best examples of the present method of treatment by mechanical agitation is the slime plant at the Bonanza mine. The following description of the treatment is compiled from the very full particulars kindly placed at our disposal by P. S. Taverer.

The vats used for collecting the slime and dissolving the gold are 24 ft. diam. and 8 ft. deep. The underflow from the slimes spitzkasten is allowed to run into one of these vats for eight hours, in which time the quantity of solid slime collected is about 25 tons. Clear water overflows during the filling, and decantation through a jointed pipe begins as soon as the tank has received sufficient slime, and continues for three hours, during which period the settlement of slimes is completed. The 1 to 1 pulp remaining is then agitated and sampled, after which 4 to 4½ tons of KCy solution at 0.008 per cent. is added for each ton of dry slime in the vat, and this occupies two hours. Then the agitator is set to work, and in addition to this the pulp is also circulated by a centrifugal pump, which draws from the bottom of the vat and delivers over the top of it. This combination of stirring and circulating for

dissolving the gold is continued for a period of about eight and a half hours. Then the pulp is pumped to one of a second set of vats placed above the level of the precipitation boxes. This pumping occupies one and a half hours, and half an hour is allowed for sweeping and cleaning the first vat for the reception of a new charge. Thus the total time required in the first vat is twenty-four hours, and three vats are just sufficient, although it is, of course, better to have some margin to provide for accidental delays. In the upper vat, which is 24 ft. by 10 ft., the pulp is allowed to settle completely, and the solution is decanted off to the precipitation boxes in which the zinc-lead couple is used. This operation requires twenty hours, after which the stirrer is started again and wash solution added in the ratio of 4 tons to 1 ton of slime, which occupies two hours. Nine hours are then allowed for partial settlement and decantation, and one hour for transferring the partially settled pulp to the final settlement tank, which is full of solution, and large enough to give a clear solution overflow as the pulp is delivered into it below the surface level. This overflow goes to a new charge of slime in one of the dissolving vats. The total time occupied in the upper vats is therefore thirty-two hours, so that four of these are required. The actual recovery of gold obtained at this mine is officially reported at 83 per cent.

In many of the most recent slime plants on the Rand no mechanical stirrers are provided, and the method introduced by J. R. Williams is adopted, namely, the sole use of centrifugal pumps to give the necessary agitation, whereby very large tanks can be used. Vats 50 feet in diameter, with sides 12 ft. deep, and with conical bottoms, whose centres are 3 ft. 6 ins. lower than their outside edges, are commonly used. The slime water is run into these at the rate of about 100 cubic feet per minute through vertical shoots, dipping about 18 ins. below the overflow level of the vats. When a vat is full, a clear water overflow is obtained, so that the filling can go on continuously until the required quantity of solid slime, say 200 tons, is collected in the vat, when the slime water is diverted to the next empty vat. After the usual settlement and decantation, a central valve in the bottom of the vat is opened, the outflow pipe therefrom being connected to a centrifugal pump. A stream of KCy solution, under pressure produced by another centrifugal pump, is delivered through a nozzle on to the mass of settled slime, which is thereby slowly hydraulicked into the outflow pipe, and the mixed slime and solution are transferred by the first mentioned pump to another similar vat, and circulated until the gold is dissolved. Then the processes of settlement, decantation, and hydraulicking are repeated, wash solution being used, and finally the residues are hydraulicked out with clean water, and pumped to the slime dam.

By using the dissolving solution for two successive charges of slime before precipitation, a smaller bulk of solution has to pass through the boxes, and the cost of this part of the work is thereby somewhat reduced. Also a smaller precipitation plant may be used, or, on the other hand, with a given plant, a more perfect precipitation may be obtained. An example will show how this

works out in actual money values, and for this purpose we will take the case of slime carrying 3 dwts. of soluble gold, and containing its own weight of water, treated with 8 tons of added solution in two operations. The value of precipitated solution is assumed at 3 grains per ton. Then, by the ordinary treatment method, the gold left in residues is 5·76 grains per ton. But by treating two charges with the same first solution, the figures are as follow:—From the first charge, of course, the same result is obtained, but the solution is worth 16·8 grains of gold per ton, when it goes on to the second charge, which, added to the value of the fresh slime, makes the total value equal to 139·2 grains per ton of slime, of which 27·84 grains will remain after the first decantation. The next solution wash at 4 to 1 adds 12 grains, making total 39·84 grains, so the final residue of the second charge is 7·97 grains as against 5·76 grains by the ordinary method. The loss in gold is therefore 2·21 grains = 4·42 pence, which is less than the cost of precipitating 4 tons of solution, and consequently there is some net gain by using this means of enriching the solution before precipitation, in the case of low grade slimes. By this method an actual recovery of 60 to 70 per cent. is obtained from slimes ranging between 2 and 4 dwts.

**Other examples of Natural Settlement Method.**—Similar work has been done on a smaller scale in the United States, as shown by the following outline of the treatment at the Schmidt & Johns mill, Silver Star, Montana, given by Matt. W. Alderson,\* who built the plant for treating the usual mixture of sand and slime obtained after the pulp has been originally caught in tailings dams. The agitating vats are each 16 ft. diam. by 8 ft. deep, having a central vertical shaft, which is supported above the vat and carries two propeller-shaped stirring blades. This vat will treat 12 tons of compact slime per charge. A twelve hours' agitation is required for the satisfactory dissolution of the gold with 0·06 per cent. cyanide solution. Two other washes are given, but the last one does not pass through the precipitating box, and the quantity of solution used for each wash is said to be from 1 to 2½ tons for each ton of slime. It is evident that the material must have varied considerably, probably owing to more or less sand being present. Later, this and other similar plants in the district have been used for treating slimes received directly from the amalgamation and concentration apparatus. Lime has been used throughout to assist settlement, and it is of interest to note that filtration through a filter formed of sand was tried at first, but was given up in favour of decantation.

Slimes are treated by the same method at several mines in Victoria by the Deeble patent agitating vat, which has an overflow gate that can be lowered and raised as required. This apparatus, as used at the South German mine at Maldon, which, according to W. B. Gray,† is 18 ft. in diam. and 4 ft. deep. It is filled to a depth of 2 ft. with cyanide solution, then the stirring apparatus is started, and 15 tons of slime are gradually charged in, thus bringing the

\* *Eng. and Min. Jour.*, vol. lxxvii. p. 757.

† *Trans. Aust. Inst. of Min. Eng.*, vol. v. p. 138.



mixture to about 3 ins. from the top of the vat. The solution is then made up to a strength of 0.15 per cent. KCy, and the pulp is agitated for at least thirty hours. Next the stirrer is stopped and raised and the pulp is allowed to settle for eight to sixteen hours "according to fineness." The words quoted show that the material treated is not always pure slime. The decantation of clear liquor is effected by depressing the overflow gate. A second solution is then applied, which is agitated for thirty minutes, allowed to settle, and the clear liquid decanted.

It would appear from the above figures that the slime must have contained something like 30 per cent. of moisture when put into the vat. Allowing for this, the dissolving solution would only amount to about 1.3 tons per ton of dry slime, and this would form a very thick pulp, which possibly accounts for the very slow dissolution of the gold (see page 216). The following figures show how the time necessary for dissolution was determined.

After 8 hours' agitation, gold in solution was 2 dwts. 0 grs. per ton.					
" 16	"	"	4	" 13	"
" 24	"	"	5	" 21	"
" 30	"	"	6	" 12	"
" 36	"	"	6	" 12	"

**Continuous Settlement Processes.**—Attempts have been made to work out a continuous method of treatment, by passing the slime pulp through a series of conical vats which are intended to act as spitzkasten.

The arrangement devised by E. T. Rand\* will illustrate this proposed method. The overflow from the sand settling vats is thickened by a spitzkasten, or by a conical vat of such a size that the amount of pulp fed into it would take three hours to fill. The clear water overflow from this goes as usual to the reservoir, while the thickened underflow is sent to a dissolving vat with a conical bottom, and on its way to this vat the pulp passes along a launder where KCy is added to it. From the launder a pump delivers the pulp to a Butters Distributor at the top of the dissolving vat. The same pump by a separate pipe, at the same time, draws pulp from the lowest point of the vat and delivers it to the same distributor; so that this pump not only supplies pulp to the vat, but also continuously circulates it. A second pump continuously withdraws pulp from the dissolving vat at the same rate as it enters and sends it to a conical vat (having a cone angle of about 60°), which, for convenience, may be termed the first settler. The clear solution overflow from this passes to the precipitation boxes, and the thickened underflow, after being again diluted with wash solution, passes into a second conical settler. The clear overflow from this one is strengthened by the addition of KCy, and is used to dilute the fresh pulp, which is then on its way to the dissolving vat. The underflow from the second settler carries away the treated slime.

By means of a roughly improvised plant on this principle about 1000 tons

\* *Jour. Chem. and Met. Soc. of S. Africa*, vol. ii. p. 103.

of slime were treated at the Simmer & Jack mine near Johannesburg. The results of the trial given by the inventor of the process are as follow :—

Original average value of slime, 1·8 dwts.

Certain selected residues, 0·3, 0·15, 0·15, 0·40, and 0·5 dwt. per ton. The average of these, which are probably some of the best, is 0·3 dwt. Calculated on this basis, the theoretical extraction is 83 per cent. Only one washing could be given in this experimental plant, and it is to be noted that the actual recovery of gold, the total quantity of solution handled, and the amount lost in the final effluent are not stated.

The Clancy continuous process is almost identical in principle, and the apparatus proposed for it is very similar. We may briefly compare the continuous and intermittent methods in the following way :—

1. The capacity of the dissolving vats must be the same in either case, because this is determined by the time required for dissolution.

2. With the settling vats in the intermittent process, a certain amount of time is lost in filling with liquor and re-pulping after each settlement, and from this point of view the total settling capacity in the continuous plant could be about 10 per cent. less than with intermittent working.

3. But as there is always some disturbance in the settlers in the continuous process, it is extremely unlikely that a 1 to 1 effluent could be obtained, even for a time, with a total settling capacity equal to that of an ordinary plant. Then, again, it is difficult to regulate the outflow of the thickened product from each conical settler to such a nicety that the maximum concentration would be maintained. Consequently the average moisture in the slimes after each operation will be greater than at the same stage of the intermittent treatment, and the gold extraction will be correspondingly less.

**Travelling Filters and Vacuum Chambers.**—A semi-continuous process was patented in 1893 by W. Brunton, in which, after dissolution of the gold in the ordinary way, the pulp is poured on to a travelling endless belt composed of filter cloth, which passes over vacuum chambers, whereby the moisture is sucked out and the dried slime then carried away on the belt and discharged.

More recently, the same idea has been adopted by T. T. Draper, of Melbourne, who has patented a machine having a rotating drum with a series of peripheral vacuum chambers. A belt of filtering fabric, suitably supported, passes over a portion of the outer face of the drum, and each vacuum chamber is connected to the exhausting apparatus only while it is in contact with the belt. So far as we know, neither of these plans has yet been tried on a practical working scale.

## CHAPTER XXXIII.

### SLIME TREATMENT WITH FILTER PRESSES.

WE have already referred to the advantage gained by reducing the residual moisture in the slimes by filtration, and for this purpose the press is undoubtedly the most effective apparatus, because it affords a very large filtering surface in a compact and handy form, and also gives a product containing not more than 20 per cent. of moisture. The following table shows the percentage of dissolved gold extracted, calculated on this basis.

TABLE LII.—*Percentage of Dissolved Gold extracted when  $m = 0.25$ , that is, when Pressed Cake contains 20 per cent. moisture.*

Value of $a$ .	Value of $n$ = number of times pressed.	
	1	2
1	75	84
$1\frac{1}{2}$	80	88.89
$1\frac{3}{4}$	83.33	91.84
2	87.5	95.06
$2\frac{1}{2}$	90	96.70
3	91.67	97.63
$3\frac{1}{2}$	92.86	98.22
4	93.75	98.62
5	95	99.09
6	95.83	...
7	96.43	...
8	96.88	...
9	97.22	...

A comparison with the corresponding figures in Table XLIX. shows at once the enormous gain in extraction due to filter pressing, this being most marked when small quantities of solution are used. On the Witwatersrand it is generally considered that 4 tons of solution per ton of slime is the minimum quantity necessary to dissolve the gold rapidly. With this amount, one pressing gives 93.75 per cent. against 75 per cent. from one decantation. Or making the comparison in another way, a total of 4 tons of solution with presses gives exactly the same result in *one* operation, as we get from 7 tons of solution with *two* decantations. This, however, does not represent the full

advantage of filter pressing from the extraction point of view, because part of the 20 per cent. of moisture remaining in the cakes can be recovered by a much smaller amount of wash liquor than is necessary in the decantation method. So that although we have shown in Table LII. the results obtained by twice pulping and pressing for the purposes of comparison, it may safely be said, when presses are used, that twice pulping with KCy solution will never be required, because the washing can be more effectively carried out on the cakes in the press.

**Methods of Filter Pressing Slimes.**—Several methods of treating slime in connection with filter presses have been evolved in Western Australia, owing chiefly to changes in the character of the ore.

1. In the first method, after the sands and concentrates have been eliminated from the mill pulp, the slime-bearing water is sent at once through the presses, whence it issues quite clear, while the slime remains in cakes in the press, and there receives the whole of the cyanide treatment for dissolving the gold as well as extracting it. This method was chiefly used with oxidised ore, and is only suitable when the gold can be rapidly dissolved, otherwise the number of presses required will be very great. In some cases, intermittent aeration, by the passage of compressed air into the press, was employed in working this system.

2. In the second method, which is the one most largely employed at the present time, the slime pulp is first allowed to settle in vats and the clear water is decanted off. The thickened pulp, then about 1 of slime to 1 of water by weight, is treated with KCy by agitation with mechanical stirrers, and is then sent to the filter press for the extraction of the gold-bearing solution, after which the pressed cakes are washed and discharged. In this method a smaller capacity is required in the presses: for instance, the first method was formerly employed at the Lake View Consols mine with ten presses,\* but at the present time the second method is practised with only four presses.

The chief objection to this system is the accumulation of solution which must occur unless a large quantity of weak liquor is run to waste. The 1 ton of water mixed with each ton of slime when first settled is increased by the addition of KCy solution to at least  $1\frac{1}{4}$  tons of liquid. As only  $\frac{1}{4}$  ton of moisture remains in the pressed cake, it follows that there is 1 ton of solution extracted from each charge, of which only  $\frac{1}{4}$  ton is required for use again with a following charge, therefore  $\frac{3}{4}$  ton of solution per ton of dry slime remains to be disposed of in some way or other. If this could be economically exhausted of its gold, as for instance by electrical precipitation, it could then be run to waste without serious loss, but with a few grains of gold per ton remaining, the loss becomes comparable with that which is due to the residual moisture in the decantation process. It may be remarked here that the difficulty of dissolving gold in thick pulp experienced on the Rand does not seem to occur to the same extent in Kalgurli. Returning to the question of waste solu-

\* H. F. Balman, *Trans. Inst. Min. Eng.*, vol. xvii, p. 359.

tion, according to Clement Dixon the precipitation of solution from the presses is very complete even with ordinary zinc shavings, as used in New Zealand, where the filter-pressed liquor never contains more than 4 grains after precipitation, and usually not more than  $1\frac{1}{2}$  grains per ton. But in order to avoid even this slight loss, a third method of working has been introduced.

3. In this case the pulp is passed first into a press to expel the water, then the cakes are discharged from the press and re-pulped in an agitation vat with KCy solution, and finally pressed a second time and washed. By this method no surplus solution has to be dealt with, because the slimes contain the *same* percentage of moisture when the treatment is finished as they did when first

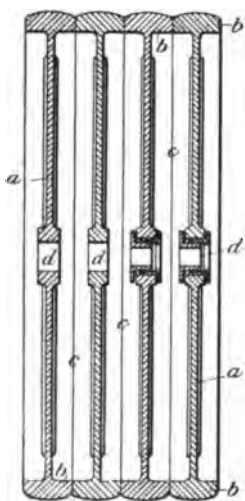


FIG. 50.—Chamber Press.

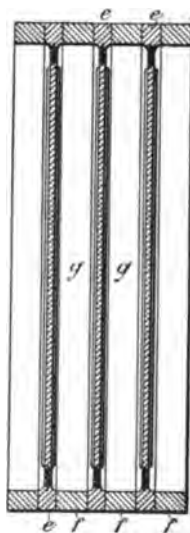


FIG. 51.—Frame Press.

brought into contact with the cyanide solution. On the other hand, the press capacity required is practically double that of the second method.

4. A slight modification of the third method has been patented by S. H. Johnson and H. L. Sulman, who propose the following series of operations. Settlement of slime and decantation of clear water, followed by filter pressing the thickened pulp. Displacement of water remaining in the cakes by cyanide liquor of normal strength. The cakes are then discharged on to a cement floor to avoid loss of solution which would already contain gold, and they are then re-pulped and treated as in method No. 3. The advantages of this method are not obvious, while the risk of losing rich gold-bearing solution in the first press, by reason of some cakes being more permeable than others, is very great. The use of KCy in the first press is also likely to cause loss of gold in the water pressed out of succeeding charges.

**Types of Filter Presses.**—In figs. 50 and 51 part sections are given of two ordinary types of press known as the “Chamber” and “Frame” press respectively.

The Chamber press consists of a number of plates *a*, corrugated on each face and provided with projecting flanges *b*, so that when the plates are pressed together, the spaces *c* within the flanges constitute a series of closed chambers for the reception of the material to be filtered. In this type the plates usually have a central aperture *d*, so that there is a continuous central channel through the press in free communication with each filter chamber. It is necessary to have a filter cloth on each side of each plate, with a corresponding central hole. In some presses these are fixed in the following manner. Two cloths are sewn together all round these centre holes; one cloth is then pushed through the hole in the plate and opened out, then both are hooked on to the top of the plate.

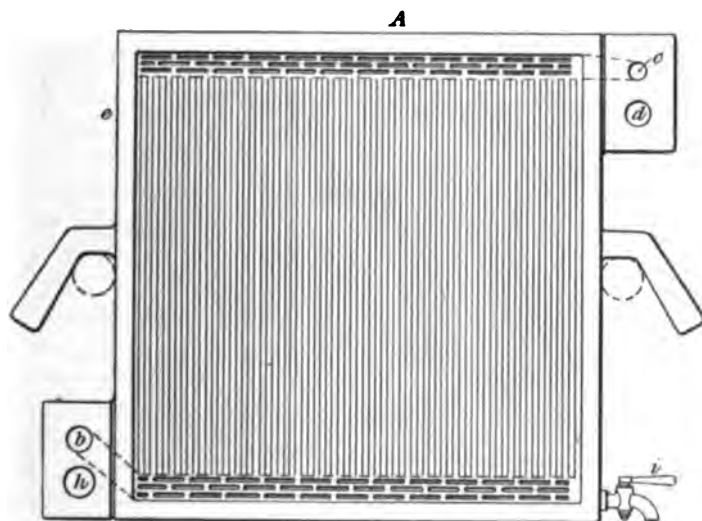


FIG. 52.—Dehne Press.

In the Dehne press the filter cloth is hung over the top of the plate so that it hangs down on each side, and is fixed by a flanged bush on each side which screws into the hole of the plate, the cloth being thereby gripped between a boss on the plate and the flange of the bush.

The Frame press (see figs. 51 and 52) consists of corrugated plates *e* and separate frames *f*, placed alternately so that the latter enclose the filtering chambers *g*. In this type also the filter cloths are simply hung over the plates and require no fixing at the centre, because the channel or channels are formed either in projecting lugs or in the margins of the plates and frames. The cloths themselves constitute the packing between the planed margins of plates and frames. The joints in the channels in the lugs are made good by filter cloth bags slipped over the lugs, or by cloth-covered india-rubber rings placed in a recess formed round the holes. Each plate is also usually provided

with an efflux tap for the discharge of filtered liquor, and when it is not necessary to discharge under pressure, the use of these separate taps, instead of a discharge common to all the plates, is preferable. This enables the operator to see whether the filtration is satisfactory from each individual chamber, so that he can shut off any chamber which is not filtering properly.

Frame presses giving cakes 2 inches or 3 inches in thickness are generally employed in the treatment of slimes, each press having usually 50 chambers with plates 40 inches square. Larger ones have been used, but are not recommended by the makers. A press of the size mentioned with 2-inch frames holds about 3 tons, and with 3-inch frames about  $4\frac{1}{2}$  tons, when full of pressed slime weighing 130 lbs. per cube foot. As the 3-inch press costs only 12 per cent. more than the 2-inch, it is evidently cheaper to use the former

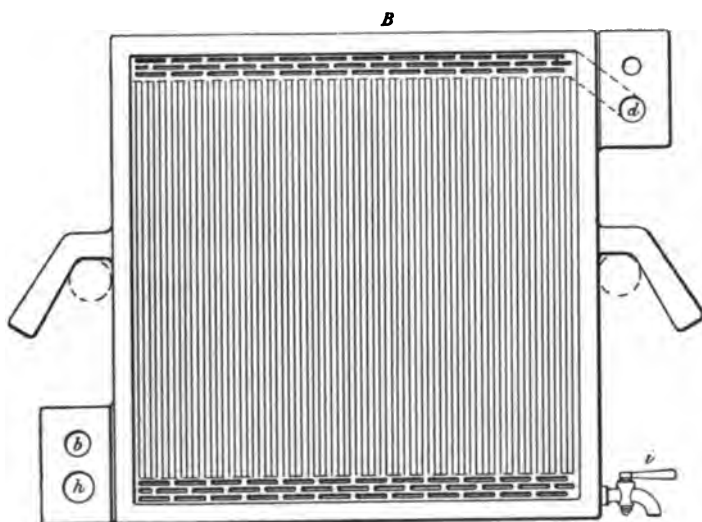


FIG. 53.—Dehne Press.

whenever the material will allow of this without unduly retarding the speed of filtration, because an economy is effected in first cost as well as in the work of emptying. It may be mentioned here that for pressing gold sludge from precipitation boxes, presses giving 1-inch cakes are generally used.

In forcing the pulp into the chambers the pressure may be increased, often with advantage, as the cakes get thicker, but it should always be steady; therefore when pumps are used for this purpose, a large air vessel should be provided to prevent fluctuation.

It is more customary, however, to fill from a montejus, which is simply an air-tight vessel, usually cylindrical, with dished ends, provided with a valve at the top for filling it with pulp. It has also an inlet for compressed air at the top and a pulp outlet pipe, which extends nearly to the bottom of the montejus at its receiving end, and is connected at its other end to the feeding channel

of the filter press. The capacity should be sufficient to hold the pulp required to fill one press with cake. For example, if the pulp contains  $1\frac{1}{2}$  parts of water to 1 part of slime, the montejus for a  $4\frac{1}{2}$ -ton press should contain 3.6 tons dry slime, at 12.2 cubic feet per ton, = 44 cubic feet plus 5.4 tons of solution at 32 cubic feet, = 173, or altogether 217 cubic feet. Under these conditions one montejus is necessary for two presses. Pressures from 45 to 100 lbs. per square inch have been used for forcing the slime pulp into the presses.

The Dehne press has been so largely used in Western Australia that a description of the arrangements provided for making and washing the cakes cannot fail to be of interest. Two kinds of plates are used, which, for brevity, we may call *A* and *B*. It will be seen from fig. 52 that the corrugated surfaces of the plate *A* are connected to an inlet channel *b* and an air escape channel *c*. In plate *B* (see fig. 53) the surfaces are not connected to either of these channels, but are open to the solution outlet channel *d*. These plates occur alternately, as seen in fig. 54, with the usual frames *f* between them carrying filter cloths as shown by dotted lines. The filling channel *h* is, of course, connected through the frames *f* to the spaces within them between the filter cloths. When the pulp is forced into these spaces the liquor passes through the cloths, while the solid slime forms a layer on the surface of each cloth until the whole space is filled with slime cake.

An outflow tap *i* is connected to each plate of each kind. A. G. L. Dehne states that by passing differently coloured materials through a press in succession during one filling, he has shown that the solid matter is deposited all over the filter cloths in *even* layers of *uniformly* increasing thickness. When the chambers are full the channel *h* and the taps *i* are closed, and wash liquor is supplied through the channel *b* to plates *A*, along the grooves of which it spreads and presses against the outside of the cloth over the whole area of one side only of each cake. It therefore passes horizontally through the cake and along the grooves of the plates *B* to the outlet channel *d*. The air channel *c*, which is open at the beginning of

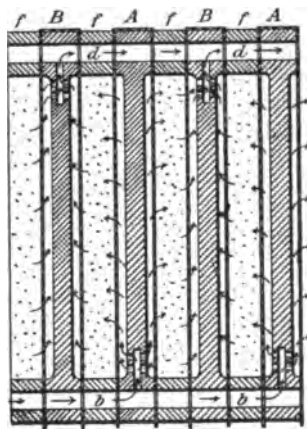


FIG. 54.—Dehne Press.

the washing, is closed as soon as water begins to flow out of it. The arrows in fig. 54, which for the sake of clearness is not drawn to scale, show the direction of the wash liquor. Instead of using channel *d*, the same result can be obtained by opening the outlet taps in the *B* plates, while those in the *A* plates are closed. After this washing is completed the press is opened and the cakes discharged.

Instead of the ordinary corrugated plate, sometimes a plate having a few ribs is used, covered with a perforated metal sheet. This gives a somewhat better support to the cloth, but it slightly reduces the effective filtering area.



Messrs S. H. Johnson & Co. recommend a plate whose surface is uniformly dotted over with little flat-topped pyramids, which result from two sets of parallel grooves intersecting each other over the whole surface.

For small presses a central screw in one end of the framing is generally used for tightening up the plates, but for large apparatus bell crank levers are employed, whose short arms press against the plates, while their long arms are drawn together by a screw, so that a great purchase is obtained.

The effluent solution from the presses is generally clarified more completely by passing it through an auxiliary press before it goes to the precipitation boxes. As no great pressure is required in this case, a lighter construction can be adopted, so that the apparatus is much cheaper than that used for the first pressing. Dehne's "Excelsior Filter," which is used on several mines in W. Australia for clarifying the solutions, is simply a cheap form of filter press in which two channels are provided for the passage of the liquor, and means are provided, by separate channels or cocks, for drawing off any liquor remaining in the chambers before the press is opened. Similar auxiliary filters for the same purpose are made by Messrs Johnson & Co.

We are pleased to acknowledge the information concerning the construction of filter presses courteously supplied by Messrs C. Harzer & Co., the London agents for the Dehne press, and by Messrs J. & R. Niven, the Johannesburg agents of the Johnson press.

**Treatment Capacity of Presses.**—With reference to the time required for filling and emptying, Wm. McNeill gives the following figures for a 20-chamber press, holding about  $1\frac{1}{4}$  tons of dry slime in cakes 28 inches square by 3 inches thick.\*

Filling press from receiver, . . .	15 minutes
Washing cakes in press, . . .	19 "
Discharging and closing, . . .	16 "
Total, . . .	50 "

Taking it at one hour, a press should accordingly treat 24 times its own capacity of dry slime per day. Or putting it another way, for each ton treated per day, the press capacity required would be 0·042 ton.

According to other published figures,† two presses, each holding 5 tons of dry slime, are sufficient to treat 120 tons daily; this makes the press capacity 0·080 ton, per ton treated daily, which is nearly double the figure derived from Wm. McNeill's data.

But from Table LIII., given below, it is quite clear that such good results as the latter figure even indicates, are seldom if ever obtained in practical operations on a commercial scale. The average capacity provided on the five mines mentioned is again nearly double that of the second estimate given above, so that in practice the average time occupied in one cycle of operations with each press is four hours. It is clear that any improvements in design which will lessen this time will be of considerable value.

\* *Trans. Inst. Min. and Met.*, vol. vi. pp. 247 to 266.

† *Jour. Chem. and Met. Soc. of S. Africa*, vol. iii. p. 59.

TABLE LIII.—*Showing Approximate Tonnage treated by Filter Presses in Western Australia.*

Name of Mine.	No. of Presses.	Thick-ness of Cake. Inches.	Total Capacity of Presses. Dry Slime.	Tons treated per day.	Press Capacity required per ton treated per day.
Great Boulder Perseverance, .	12	3	43·2	216	0·2
Associated Gold Mines, . .	8	3	28·8	220	0·131
Great Boulder Main Reef, .	4	3	14·4	73	0·197
Kalgoorli Gold Mines, . . .	3	2	7·2	46	0·157
Lake View Consols, . . . .	4	3	14·4	91	0·158
Ivanhoe, . . . . .	6	2	14·4	133	0·108
Totals and average,	37	...	122·4	779	0·157

The above table may be useful in laying out plants in other countries, as it shows the limits of variation as well as the average capacity provided in practice. It must, however, be remembered that in W. Australia the slimes filter pressed often contain ore particles in considerable quantity, which in other countries would be classed with sands. From this it follows that where a 3-inch cake is practicable in W. Australia, it may be impracticable in countries where the slimes are freer from sands. A thinner cake has then to be employed, which reduces the capacity accordingly.

**Kalgoorli practice.**—For the following particulars of several Kalgoorli plants and methods of working we are largely indebted to a pamphlet by Robert Allen, M.A., B.Sc., to which we have previously referred. At the first three mines mentioned the whole of the ore is reduced to slime for cyanide treatment.

At the Great Boulder Main Reef the sulphide ore is broken by a Gates' Crusher to  $1\frac{1}{2}$  inch, and then goes through Krupp ball mills with No. 30 screens, after which 80 per cent. of the product will pass a 100-mesh. The ore is roasted in mechanical furnaces, which discharge into a launder in which a thin pulp is formed by the addition of weak cyanide solution. A tailings wheel lifts this pulp to a series of spitzkasten, which give a clear water overflow and separate out two products, namely, sands and slimes. The latter are pumped up to agitator vats 21 ft. diam. and 6 ft. deep, where more KCy is added and the pulp is stirred for about eight hours. The sands are amalgamated and ground fine in Wheeler pans, and the continuous overflow from these is again separated into sands and slimes. The former are returned to the pans and the latter are sent to the agitator vats above mentioned, which ultimately receive the whole of the material crushed. When the gold is dissolved, the pulp is run into the montejus and thence forced by compressed air into the filter presses, whose effluent passes through an auxiliary press to the zinc boxes. Of the whole gold recovered on this mine 90 per cent. is said to be

obtained in the cyanide plant, the other 10 per cent. being presumably caught by the mercury in the Wheeler pans.

At the Associated Gold Mines the treatment is very similar, the principal difference as regards cyaniding being that, whereas in the method just described the KCy is added before the fine grinding and amalgamating, at this mine water is used in the pans, for the reason given below, and the cyanide solution is added afterwards. The sulphide ore is broken to 2-inch pieces in Comet Breakers, then to 1 inch in Roger Rolls, next crushed to 30-mesh in ball mills, and roasted in mechanical furnaces. After this the ore is mixed with water and passed through grinding and amalgamating pans and so-called settlers, each with a continuous overflow. From the settler overflow the pulp is pumped to settling vats 35 ft. diam. and 8 ft. deep, from which clear water is decanted. By this means the ore is well washed, and the 'cyanicides' removed, before coming into contact with the cyanide. The thick settled pulp is passed to agitator vats 18 ft. diam. and 5 ft. deep, where it is stirred up with solution until the gold is dissolved. It is then filter pressed in the usual manner.

At the Hannan's Star mine the sulphide ore is treated, without roasting, by the Diehl process, of which we give the following outline. After going through a Blake-Marsden Crusher and Krupp ball mills with 30-mesh screens, about 65 per cent. of the material will pass a 100-mesh screen. This product is divided into sands and slimes by spitzkasten, and the sands, after passing over amalgamating tables, are slimed in a Krupp flint mill (see below). All the slimes are treated together in agitator vats with KCy and CyBr for about twenty-four hours, and afterwards filter pressed. The effluent solution passes into a settling vat, whence it is syphoned over to a filter vat, and from the latter passes to ordinary zinc precipitation boxes.

H. Knutsen \* gives a very complete account of the Diehl process as carried out at Kalgurli. After the ore has been crushed in one of the usual ways, it is re-crushed in a Krupp-Grusonwerk grit or flint mill. This has a cylinder about 18 feet long by 4 feet diameter, and contains about 4 tons of flints. The sand from the crushing mill is fed with water, until the mass contains 3 to 5 per cent. of solids, through a hollow journal at one end of the cylinder, and escapes through another hollow journal at the other end, as the cylinder rotates. The slimed product leaving the mill flows on to a set of classifiers, where the sands are separated, to be returned to the mill for re-crushing. The slimes leaving the classifiers are thickened until they form a pulp of 40 to 50 per cent. of solids. This is then transferred to agitators, which consist of covered vats 20 to 25 feet in diameter and 7 feet 6 inches to 8 feet deep, their capacity being 100 to 125 tons of pulp.

For slimes containing 1 to 3 ozs. of gold per ton, a strong solution of cyanide is added, equal to 4.4 lbs. KCy per ton of solids. After agitating for 1 to 1½ hours, bromide of cyanogen in solution is run in, about 1.1 lbs. per ton

\* *Min. Jour.*, July 26, 1902, p. 1024 *et seq.*, from a paper read before the Institute of Mining and Metallurgy, London.

of solids. The agitation is kept up for 24 hours. For richer slimes further quantities of KCy and BrCy are added at intervals of 6 to 8 hours, but for poorer slimes smaller quantities than the above are used. About 2 hours before agitation is complete, 1 lb. to 4 lbs. of lime per ton of dry slime are added.

The pulp next flows direct from the agitator to a receiver, to be filter pressed, which holds just enough to charge the press with  $4\frac{1}{2}$  to 5 tons of solids. It is forced into the press by means of compressed air, which at the beginning of the operation has a pressure of 30 lbs. per square inch, but as the filter press fills this is gradually increased to 50 lbs., and even to 75 lbs. When the charge is complete the filter press valve is closed, and the compressed air is shut off from the sludge receiver, to be next directed to the weak solution receiver, and finally to the water wash. A current of air at a pressure of 80 lbs. per inch is lastly blown through the cakes for 10 to 15 minutes to dry them, after which the press is opened and the cakes are discharged into trucks. The press is then cleaned and got ready for the next charge. The whole time required for a complete set of operations is 2 hours.

About 35 to 50 tons of weak solution and a similar quantity of water wash is required for each 100 tons of dry solids. With poor slimes of 10 dwts. per ton the use of a weak solution is omitted. The gold-bearing solution flows from the filter press to a storage tank, to be clarified by passing through a filter at its bottom, or sometimes through a small filter press, before entering the zinc box.

## CHAPTER XXXIV

### DESIGN AND CONSTRUCTION OF VATS.

#### *Section I.*

**Vat Capacity in Cyanide Plants.**—When the nature of the material to be treated and the method to be adopted have been determined by preliminary tests or by previous work in the same locality, the total vat capacity required in the plant for treatment is easily arrived at by multiplying the daily tonnage, expressed in cubic feet, by the total number of days occupied in the treatment of one charge, including, of course, the filling and emptying of the vat. Thus, for instance, in a single treatment sand plant, if one day is allowed for filling, one for discharging, and five for treatment, the calculation is as follows: for 500 tons a day of material occupying say 24 cubic feet per ton,

$$(a) \quad 500 \times 24 \times 7 = 84,000 \text{ cub. ft.}$$

When collecting vats are to be used, then, on the basis of one day to fill, one to drain, and one to empty, the capacity required at 20 cubic ft. per ton for direct filling is as follows:

$$(b) \quad 500 \times 20 \times 3 = 30,000 \text{ cub. ft.}$$

For a double treatment plant, example (a) would be suitable for the second series of vats. For the collection and first treatment vats, allowing two days for filling and discharging and two for preliminary treatment, we have:

$$(c) \quad 500 \times 20 \times 4 = 40,000 \text{ cub. ft.}$$

This will be sufficient if the plant is so arranged that the contents of any collecting vat can be transferred to any vat of the second series as in figures 34 to 37, but with double-tier plants (figs. 41, 42) the same time must be allowed in reckoning the capacity of each set, and the only difference in the respective sizes is due to the difference in the volume occupied by the material in the upper and lower vats.

In the case of slime plants, we have not only to allow for the volume of the slime, but also for the solution used with it. Thus, if a 4 to 1 pulp is to be used in the agitation and settling vats, the total capacity is found in the following way: for, say, 100 tons of dry slime daily, with a total treatment time of 72 hours:

$$\begin{array}{rcl} 100 \text{ tons slime at } 12\cdot3 \text{ cub. ft.} & = & 1,230 \text{ cub. ft.} \\ 400 \text{ tons water at } 32 \text{ cub. ft.} & = & 12,800 \text{ ,,} \\ \hline \text{Total volume treated daily,} & & 14,030 \text{ ,,} \end{array}$$

which, multiplied by three days, is equal to 42,090 cub. ft.

The above simple calculations are sometimes useful in rough estimations, for the purpose of comparison of plants for different methods of treatment, and also for checking the results arrived at from the considerations mentioned hereafter.

For solution vats the total storage capacity usually provided in sand or slime plants is between one-fifth and one-tenth of the total treatment-vat capacity.

**Number and Size of Vats.**—As the time of filling and emptying is necessarily included in the above figures, it follows that the total capacity will to some extent finally depend on the number and size of the vats.

For this and many other reasons in designing a plant, instead of calculating the total capacity, it is better to select first of all the size of the vat, and then to decide upon the shape, *i.e.* the relation between diameter and depth, which is most suitable for the particular material in each case.

For ordinary leaching vats the most convenient arrangement is to make each vat contain the quantity treated per day, because then every operation in the cycle of treatment occurs at the same hour each day, and consequently the utmost simplicity in working and freedom from mistakes is thereby ensured. Also, by having few vats, the number of pipes and cocks for solution is kept as small as possible, and this, again, simplifies the working of the plant. As there is no difficulty in constructing vats to hold 1000 tons, or more if required, there is no excuse for designing plants with a multiplicity of small vats. The only valid objection to large vats is that, under given conditions, each takes longer to fill than a small one, and is therefore put out of action for a longer period, so that the total capacity required in the plant is thereby somewhat increased; but against this may be set the undoubted fact, that a few large vats are much cheaper than many small ones of the same aggregate capacity.

Collecting vats also can generally be designed to receive one day's output from the mill, but, as mentioned in describing their use, there are sometimes reasons for making them smaller, and the objections to small dimensions in this case are not so serious, because the only operations to be performed are filling, draining, and emptying.

When treating slimes by agitation there are self-evident mechanical reasons for not making the vats very large. In this case the time of filling is often a large portion of the total time occupied per charge, and therefore the objection mentioned above becomes important, and the cheapness of large vats is somewhat discounted. Each vat should, however, be large enough to give a clear water overflow when receiving the whole slime product from the mill, and need not be larger. Suppose, as an illustration of this point, that the slime product from a 40-stamp mill will give a clear overflow from a 24-foot vat which fills in 6 hours, and that the total treatment under these conditions takes 60 hours. In order that the time required for settlement may be constant, we may assume that the diameter only is altered, so that the vat will receive a charge equal to 24 hours' supply from the mill. Then

the total treatment time will be increased to 78 hours or more, and the total capacity required will be increased in the same proportion, that is to say, by 30 per cent. If, on the other hand, the depth is increased, the settlement time will also be lengthened, and the necessary capacity still more seriously increased. The following table has been worked out from the results of actual practice on the Rand.

TABLE LIV.—*Sizes of Vats giving Clear Water Overflow when receiving Slime Pulp.*

Diameter of Vat.	Cubic feet of Pulp received per minute.
20 feet	18
25 "	25
30 "	34
35 "	45.5
40 "	60
45 "	78
50 "	100

Having determined the diameter according to the above considerations, the depth may be about 7 to 9 feet for agitator vats, and is often as much as 12 feet for treatment by pump circulation. It is convenient to adjust this dimension so that the charge of the vat will be equal to a simple fraction of one day's supply, such as one-third or one-half.

The tendency at the present time in all departments of cyanide plants is to provide a greater capacity than formerly, in order to allow of a longer treatment and a consequently higher extraction, as the economical limit in this direction has not yet been reached.

**Most economical Shape of Vat.**—When only the cost of the vat itself is taken into consideration, it has been shown mathematically, for a vat of uniform material and thickness throughout, that the most economical shape is obtained when the depth is equal to the radius. But the cost of foundations has to be taken into account; and as this item also increases with the diameter, it follows that the depth should be greater than the radius, in order to get the greatest economy in construction. Suppose, as a simple example, that the cost of foundations per square foot covered was exactly equal to that of the vat bottom. Then the cheapest vat would be one in which the depth is equal to the diameter. But in most practical cases, and certainly where the vats are elevated on columns and girders, the supports will cost more than the vat bottom, so that theoretical economy in construction demands a depth greater than the diameter. It is evident therefore that, as a rule, the most economical shape cannot be used in practice, as other considerations, such as effective percolation and convenience in working, have more importance than a small saving in first cost. Solution storage vats can, however, generally be made

much deeper than leaching vats; and as their foundations are seldom costly, the best depth can often be used, this being something between one-half and three-quarters of the diameter. In all cases it is useful, when designing a plant, to remember in which direction economy lies.

**Pressure in Vats.**—Small vats are usually designed according to precedent, and calculations are seldom necessary, but for large sizes the stresses should be carefully determined and allowed for.

When a vat is filled with sand and water the whole mass forms what is practically a quicksand, and, so far as its pressure is concerned, it may be considered to act as a fluid, whose weight is equal to that of the sand and water together. Referring to page 55 we may take the two extreme cases there given to find the limits of this weight for quartz sand.

Mixed sand, one cub. ft. =	100 lbs.
Water, .892 cub. ft., say	24 „
Total,	124 „
Classified sand, one ft. =	80 „
Water, .52 cub. ft., say	38 „
Total,	118 „

In the following examples we shall take 120 lbs. as the weight per cubic foot of the fluid mass. For pyritic concentrates a higher figure should of course be used, which can be found in the same way.

The pressure in pounds per sq. ft. upon the side of the vat at any depth will be found by multiplying the depth in feet by 120. The figure so obtained multiplied by the diameter in feet will give the total pressure tending to burst the vat at the depth selected. This, of course, is zero at the top, and reaches its maximum at the bottom.

**Design of Timber Vats.**—In a wooden vat there is an additional stress on the hoops, due to the swelling of the staves and planks, but in the absence of exact data as to the amount of stress resulting from this cause, it is allowed for in practice by giving a little extra strength to the hoops generally, and by adding one hoop at the bottom especially to resist the swelling of the floor.

One convenient method for designing a wooden vat is to calculate the stresses separately for each foot of depth, and so determine the necessary amount of iron required in the hoops at each part, because by this means the approximate spacing of the hoops is found as well as their number and size. As only a few sizes of round iron are usually employed in vat construction, we can at once assign a value for the safe load on each size. Taking 11,200 lbs. per sq. inch, and allowing for cutting threads at the ends, we get the following figures :

$\frac{3}{4}$ in. diameter,	.	.	.	safe load is	3,405 lbs.
$\frac{1}{2}$ „ „	.	.	.	„ „	4,726 „
1 „ „	.	.	.	„ „	6,205 „
$1\frac{1}{4}$ „ „	.	.	.	„ „	7,806 „
$1\frac{1}{2}$ with $1\frac{1}{4}$ screwed ends,	.	.	.	„ „	11,133 „



Larger sizes than these are not advisable, as there is much difficulty in bringing them well up to every stave.

As an example, we will now make the necessary calculations for a vat 40 ft. diameter and 8 ft. deep inside. When working in this way it is well to draw a vertical line, such as A B in figure 55, to any convenient scale to represent the side of the vat, with the feet marked off and numbered from the top. Then the position of each hoop can be tentatively marked off on this line, and afterwards a final slight adjustment of positions can be made if desirable.

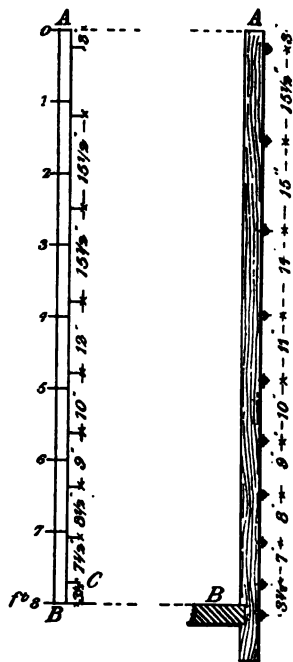


FIG. 55.

Spacing Vat Hoops.

FIG. 56.

Beginning at the bottom, the pressure at 8 feet, tending to burst the vat, is

$$120 \times 8 \times 40 = 38,400 \text{ lbs.}$$

and at 7 ft. it is 33,600 ,,

The mean of these two figures, viz., 36,000 lbs., is the total bursting pressure on the lowest foot. Half of this, namely, 18,000 lbs., is the tensile stress in the hoops supporting the lowest foot. Therefore the number of  $1\frac{1}{2}$ -inch hoops with  $1\frac{1}{4}$ -inch ends required is 1.6, and by simple proportion we find that each hoop will support a depth of  $7\frac{1}{2}$  inches. So the first hoop can be placed at C, say  $3\frac{1}{2}$  inches above the floor level, and the second hoop say  $7\frac{1}{2}$  inches

above the first. Proceeding in the same way with the next foot we have,

Bursting pressure at 7 feet, as before, 33,600 lbs.

Bursting pressure at 6 feet, . . . 28,800 ,,

Therefore, mean pressure = 31,200 lbs. and tensile stress 15,600 lbs. Number of hoops 1.38, which corresponds to a distance apart of  $8\frac{3}{4}$  inches. So the third hoop may be placed at  $8\frac{3}{4}$  inches above the second, and the fourth, say, 9 inches above the third, as in fig. 55.

This process can be continued until the calculation gives a space of more than 18 inches between the hoops, when a smaller size of hoop must be adopted, because with 3-inch staves they must be supported at least every 18 inches, however small the pressure may be. In continuing the calculation, the following figures will be obtained :

Between 5 and 6 ft.	1.18 hoop.	$10\frac{1}{4}$ ins. apart ; say, 10 ins.
„ 4 „ 5 „	0.97 „	$12\frac{3}{4}$ „ „ 12 „
„ 3 „ 4 „	0.75 „	$15\frac{1}{4}$ „ „ 15 „
„ 2 „ 3 „	0.54 „	$22\frac{1}{4}$ „ „

As the last distance exceeds 18 inches, the calculation must be repeated with a hoop of smaller size, say  $1\frac{1}{2}$  inch throughout, thus :

Between 2 and 3 ft. 0.77 hoop.  $15\frac{1}{2}$  ins. apart ; say,  $15\frac{1}{2}$  ins.

Having now risen to within 15 inches of the top, it is evident that no more calculation is necessary, and the last hoop can be suitably marked on the figure, say 3 inches below the top of the stave. Then, to improve the appearance by making the spacing of the hoops increase regularly, the distances can be adjusted as in figure 56, which also shows the extra hoop necessary round the bottom of the vat. It will be noticed that in getting rid of the small fractions of inches, and in the final adjustment, all the alterations are in the direction of increased safety.

**Graphic Method of spacing Hoops.**—The following simple geometrical method enables the number and position of the hoops to be determined very quickly. Draw a vertical line AB (see fig. 57) to represent the side of the vat to any convenient scale. At B draw a line BE at right angles to AB, and lay off BF equal to half the bursting pressure at the bottom of the vat. It is well known that the total tensile stress in all the hoops is then equal to the area of the triangle ABF, i.e. to  $BF \times \frac{AB}{2}$ .

This total stress divided by the safe load on each hoop will give the number of hoops required. Further, if we divide the triangle ABF by horizontal lines,  $i_1 k_1, i_2 k_2$ , etc., into as many equal areas as the number of hoops, then the intersections of these lines with the line AB will divide the latter in such a manner that each portion  $Bi_1, i_1 i_2$ , etc., can be safely supported by one hoop. The construction necessary for this is the following :—Bisect AB and describe a semicircle on it. Then divide AB into the required number of equal parts as at  $g_1 g_2 g_3 g_4$ .

From each of these points draw horizontal lines cutting the semicircle in  $h_1 h_2$ , etc. Then with centre A and the respective radii  $Ah_1 Ah_2$ , etc., draw circular arcs cutting AB in  $i_1 i_2$ , etc., and through these points draw horizontal lines  $i_1 k_1, i_2 k_2$ , etc., which will divide the triangle ABF into equal areas as required.

As each of these areas, for instance  $Bi_1 k_1 F$  represents the pressure sustained by one hoop, it follows that a horizontal line through the centre of gravity of  $Bi_1 k_1 F$  will represent the resultant pressure, and the point where it cuts the

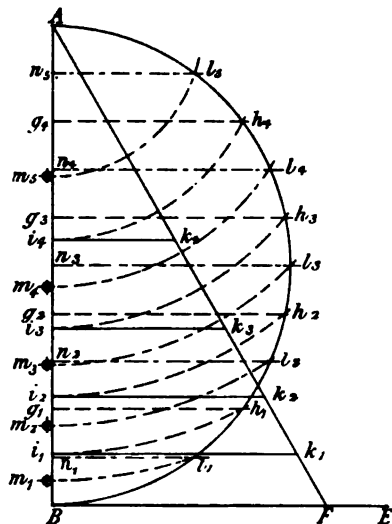


FIG. 57.—Graphic Method of spacing Hoops.

line A B will be the correct position for the hoop. But for our purpose it is sufficient, and for some reasons better, to place the hoops in the following manner, so that, considering each section separately, the pressures above and below the hoop are equal. It is obvious that this can be done by dividing A B in the first operation into *twice* as many parts as there are hoops, by means of the intermediate points  $n_5 n_4 n_3$ , etc., and by the same construction transferring these points to  $l_5 l_4$ , etc., on the semicircle, and back to the line A B at  $m_5 m_4$ , etc., which will be the required positions for the hoops. In this way the trouble of finding the centre of gravity of each separate area is avoided, and the extra strength is provided for taking the pressure due to the swelling of the staves.

Except for the purpose of clearly explaining the principles underlying this method, the drawing of the triangle and its subdivisions is not actually

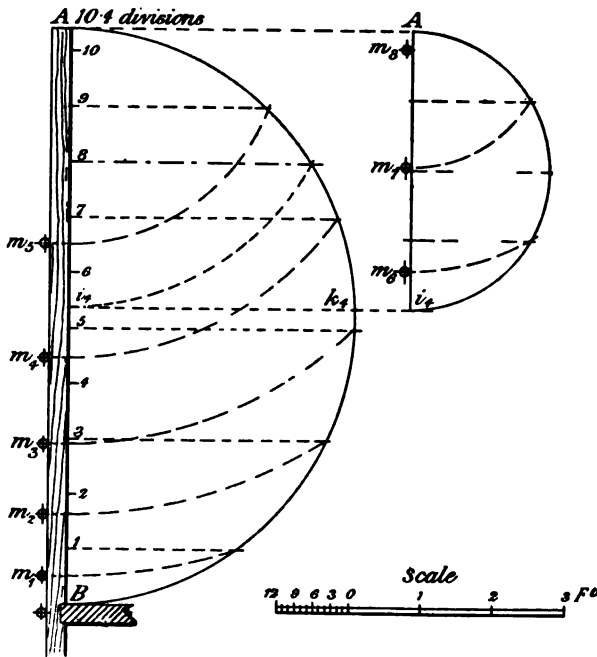


FIG. 58.—Graphic Method of spacing Hoops on a 30-foot Vat.

necessary. In the next diagram, in fig. 58, which is worked out to scale for a vat 30 feet in diameter and 8 feet deep, all unnecessary lines are omitted. The bursting pressure at the bottom in this case is 28,800 lbs., and therefore the total tensile stress in all the hoops is  $\frac{28,800}{2} \times \frac{8}{2} = 57,600$  lbs. If hoops of  $1\frac{1}{2}$ -inch round iron with  $1\frac{1}{4}$ -inch screwed ends are used, then 5.2 hoops will be required. The line A B is therefore divided into 10.4 parts and the positions  $m_1 \dots m_8$  of the hoops are found as before.

But by scaling from the diagram it is found that the distance between  $m_4$  and  $m_5$  is 19 inches, so it is better to use two hoops of smaller size in place of  $m_5$ . In fig. 58 the upper part of the stave is dealt with separately by dividing it into four equal parts, and thus finding the positions,  $m_6, m_7$ , for two hoops of 1 inch diameter throughout. In addition to these, it is necessary to put an extra hoop about 3 inches from the top. Theoretically this could be very thin, but in practice it is generally made the same size as those immediately below it, to avoid multiplicity of sizes. An extra hoop around the bottom completes the number.

As regards the floor of the vat, it is only necessary to see that the material of which it is composed is strong enough to carry this load over the spaces between the joists or other supports, but this is a matter that is usually considered in arranging such supports rather than in the design of the vat itself.

**Construction of Timber Vats.**—Vats are usually made of either red deal, Oregon pine, Californian redwood, or Kauri pine.

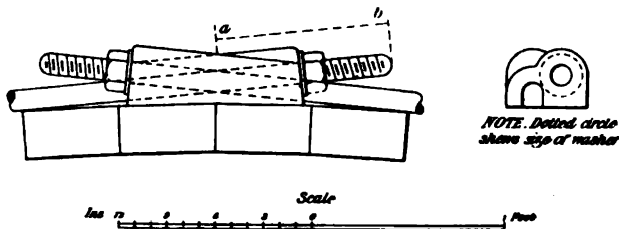


FIG. 59.—Hoop Casting.

For small clean-up vats up to 6 feet diameter, staves 4 inches wide by  $1\frac{1}{2}$  inches thick are used, with  $\frac{1}{2}$ -inch check on to bottom and 3-inch chine.

For vats up to 12 feet diameter the staves may be  $4\frac{1}{2}$  inches wide by 2 inches thick with  $\frac{5}{8}$ -inch check and 4-inch chine. Up to 25 feet, staves 6 inches wide, and for larger vats 9 inches wide, are usual, the thickness being 3 inches, check  $\frac{3}{4}$  inch, and chine 6 inches.

The staves are planed to the proper bevel on both edges by hand or machine; if by the latter, they should be hand-dressed afterwards. In first-class work the staves are also planed on both faces and thickened. The planks forming the floor of the vat are usually of the same thickness as the staves, but somewhat wider, being 6 inches for small sizes, 9 inches for medium sizes, and 12 inches for vats more than 25 feet in diameter. They should be planed in both faces and thickened, planed and hand-dressed truly square on the edges, which are generally grooved  $\frac{3}{8}$  inch wide and  $\frac{1}{2}$  inch deep, to receive clear pine tongues  $\frac{3}{8}$  inch thick and 1 inch wide.

The hoops are usually made of round iron for small sizes, because they can be screwed at the ends, whereas with flat bar, separate screwed ends have to be welded on. For large sizes it is better to weld on screwed ends whose

section inside the thread is equal to that of the bar. In some districts the saving of material in this way is greater than the cost of welding, but the main object is to avoid any unnecessary metal in the hoops, because any excess is worse than useless, as it increases the difficulty of fitting them closely around the staves. So for large hoops some prefer to use flat bar iron.

The hoops are made in several sections, which are connected by castings of the form shown in fig. 59, or in fig. 60. It is clear that each section must overlap the next one by an amount equal to  $ab$  (see fig. 59) at each end. If

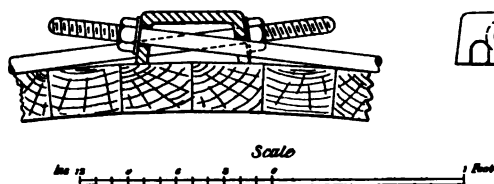


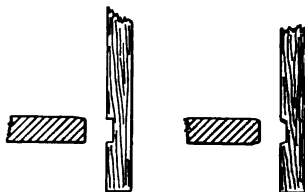
FIG. 60.—Hoop Casting.

the overlap be made too long, it means expense of extra screwing at each end, but when it is too short, there is trouble in getting on the last nut of each hoop. A good practical length is obtained by making  $ab$  9 inches for 20-foot

vats and say 11 inches for 40-foot ones. The thickness of metal in castings should be about  $\frac{1}{4}$  inch less than outside diameter of screwed ends of the rods.

The check in the stave to fit the bottom is sometimes made with parallel sides, as in fig. 61, in which case the bottom is left flat on the under side, and is only slightly bevelled on the upper side. Some makers prefer a check with inclined sides: in this case the upper and lower sides of the bottom are bevelled to the same angle as in fig. 61a. Either method gives satisfactory results, provided that the workmanship is good.

Timber vats are erected in the following manner. The joists are first placed in position and properly bedded, so that their upper edges are all truly level and in the same horizontal plane. The bottom planks are then laid down, the tongues fitted in the grooves, and the whole tightly cramped up or wedged. While cramped, cross strips are temporarily screwed down to the planks to hold the latter in place. The cramps are then removed and a circle struck out whose diameter is from 1 to  $1\frac{1}{2}$  inches larger than the inside dimension of the vat, according to the depth of the check in the staves. The bottom is then cut to this circle and bevelled round the edge to fit the checks.



FIGS. 61, 61a.—Stave Checks.

In putting on the staves it is important that each stave as it is put in place should be driven on to the bottom as far as it will go, and also close to the previous stave. For if the first point is not attended to, the circle of staves when completed will be too large, and it will then be impossible to close the staves tightly on to the bottom. If the second matter is neglected, so that the staves when placed are as close as possible to the bottom, but not close to each other, it will then be impossible to properly close up the joints between

the staves, except, of course, near the top, where they can yield to the pressure of the hoops. A little experience and care soon enables a skilled joiner to adjust matters, so that when the hoops are screwed up the staves will close tightly against each other and, at the same time, against the bottom planks. If the completed vat leaks between the staves, a thin extra stave is needed, but if it leaks round the edge of the bottom, one of the staves must be made narrower.

When all the staves are placed, a few hoops are put on and partially tightened up, after which the remaining hoops are placed in position and all are gradually screwed up.

Spanners up to 30 inches long are used, and each hoop, while it is being drawn up, is well hammered, to bring it close to each stave. The joints in the several hoops should not be placed one over another, but should be placed "en échelon," as shown in the various drawings of vats.



FIG. 62.—Plank for bottom of Launder.

Some engineers allow the abutting edges of the staves and floor planks, and also the tongues of the latter, to be smeared with a very thin layer of white lead in oil, while others prefer to rely only on the dressing of the timber for a tight joint. In Australia, the grooves and tongues in the floor are generally not used.

An excellent method of making the joints thoroughly water-tight, and which may be adopted when very rich solutions are to be dealt with, is described by J. Mactear\* as follows:—The planks to be used for staves are passed edgewise between a pair of rollers provided with a central projecting rib, under a considerable pressure obtained by weights and levers. These form a central groove  $\frac{1}{2}$  in. to  $\frac{3}{4}$  in. in breadth along each edge of the plank by compressing the fibre of the wood to a depth of  $\frac{1}{2}$  in. When the plank afterwards passes through the planing machine, the whole of the wood projecting beyond this groove is planed off to the proper bevel, so that after the tank is built and wetted, the compressed fibre expands and forms a perfectly tight joint. This method, of course, involves the waste of 1 inch of material from each stave, which is, however, not a serious matter.

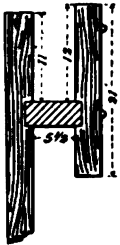
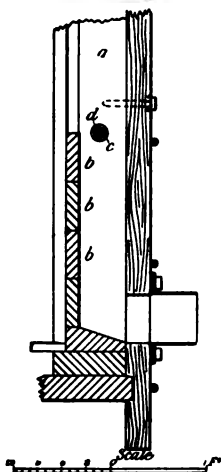
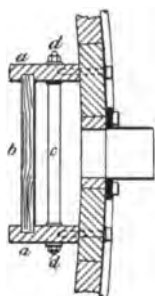


FIG. 63.—  
Cross-section  
of Launder.

**Timber Collecting-Vats.**—In these, a circular launder is formed round the outside of the vat at the top in the following manner. Each stave, before erection, is checked about  $\frac{1}{2}$  in. deep on the outside, so that these checks form a continuous groove round the vat about 5 to 12 ins. from the top. The bottom of the launder is formed from 3-in. planks cut to the shape shown in fig. 62, which are fitted into the groove just mentioned. Short staves are driven on

\* *Proc. Inst. C.E.*, 1894-5, vol. cxi. part iii. p. 22.

the outside edge of these curved pieces, and drawn together by a couple of light hoops, say  $\frac{3}{4}$  or 1 in. round iron, the result being shown in fig. 63, which is a cross section through the trough so formed.



FIGS. 64, 65.—Slat Gate ;  
plan and section.

A slat gate for a wooden vat is shown in figs. 64 and 65. Two uprights *a a*, fixed inside the staves, have grooves to receive the slats *b b*. Short pieces of pipe *c c*, with washers threaded on bolts *d d*, serve to keep the uprights from warping. The outflow orifice may be fitted with any suitable form of valve, if any cyanide treatment is to be carried on in the vat. When used for collection only, a spout cut from a 6-inch pipe can be attached to the vat by an ordinary pipe flange as shown. Suitable widths for these slat gates are—for 20-ft. vat, 1 ft. 6 ins. ; for 25-ft., 1 ft. 9 ins. ; and for 30-ft. vat, 2 ft.

**Quantities in Timber Vats.**—By the use of either of the following formulæ the quantity of timber in a vat of any size from 10 ft. to 40 ft. diam. can be quickly found. The waste of timber in cutting the circular bottoms is allowed for, and the results are correct to about  $1\frac{1}{2}$  per cent. of the total.

Let *D* = inside diameter of vat in feet

*L* = length of stave in feet

*F* = number of feet of 9 ins. x 3 ins. deals

*C* = quantity of timber in cub. ft.

*S* = sq. ft. of timber reduced to 1 in. thick

Then,

$$F = aDL + bD^2$$

$$C = cDL + dD^2$$

$$S = eDL + fD^2$$

The constants of the first terms are—

$$a = 4.35$$

$$c = 0.816$$

$$e = 9.787$$

The coefficients of the second term vary slightly with the diameter and are therefore tabulated below.

TABLE LV.—*Values of Coefficients b, d, and f.*

Diameter of Vat in feet.	<i>b</i>	<i>d</i>	<i>f</i>
10	1.210	.227	2.724
15	1.194	.224	2.688
20	1.178	.221	2.652
25	1.162	.218	2.616
30	1.147	.216	2.592
35	1.145	.215	2.580
40	1.143	.214	2.568

The same formulæ can be used for collecting-vats by adding 2 ft. 6 ins. to the actual length of the stave, to allow for the timber in the peripheral overflow trough.

For the quantity of iron in the hoops we have no simple or exact formulæ, but the one given below is within 10 per cent. of the correct weight for hoops arranged according to the methods previously given. Thus, if  $W$  = total weight of iron in lbs., then

$$W = (D + 31)^2 (L + 4)^2 \times .004$$

This looks rather complicated for the purpose, but when it is used with a table of squares, such as is found in engineering pocket-books, the amount of work involved is very much less than that of taking out the weights in detail, unless working drawings are available. The weight of the cast iron connections can be taken at one-half to two-thirds of the weight of the hoops.

**Timber in Joists.**—For deep vats with discharging tunnels beneath them, 9-in.  $\times$  6-in. joists are generally used, spaced at 2 ft. 3 in. centres, and for shallow vats, 9-in.  $\times$  3-in. joists at 2 ft. centres are sufficient. The first case, including herring-bone struts, requires nearly 1 foot of 9-in.  $\times$  3-in. stuff per square foot of tank bottom, and the latter case, a little more than half this amount. Therefore, to obtain the quantity of timber in the joists, it is only necessary to multiply the area of the bottom by some number between 1 and 0.5, depending upon the depth of the vat and the maximum width of opening spanned by the joists. The latter is usually not more than 6 ft. or less than 5 ft., and the following formulæ, with the same notation as before, will, in average cases, give an approximately correct result :

$$F = 0.0486 D^2 (L + 4)$$

$$C = 0.0082 D^2 (L + 4)$$

$$S = 0.0981 D^2 (L + 4)$$

The above empirical formulæ, and others of the same kind which will be given later, are not intended for use in close estimating when drawings have been made, but they will be found useful in making rough estimates of cost, and for getting the relative costs of vats of various sizes under similar conditions as to material.



## DESIGN AND CONSTRUCTION OF VATS.

**Strength of Steel Vats.**—In designing large steel vats, the quickest method is to use formulæ, such as those given below, to find at once the thickness of metal required at any depth. Generally, it is only necessary to make the calculation for the full depth, and to use the same thickness throughout.

*Note.*—For steel plates with single riveted lap joints, or butts with single covers, we may put  $k=8620$  lbs.

$$\therefore P = Dd\omega$$

Therefore  $t = \frac{Ddw}{24k}$  . . . . . (3)

$$t = \frac{Dd}{1724} = 0.00058 Dd \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$
$$t_1 = 0.00058 \text{ rad}$$

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but in this case with a spherical bottom curved to a 30-foot *radius*, the depth at the centre will be 14 feet and the thickness must be 0.224, or say  $\frac{1}{4}$  inch.

The following dimensions represent ordinary practice as regards thickness of plates.

Up to 25 feet diameter and 10 feet deep,  $\frac{3}{16}$  inch throughout. From 25 to 30 feet diameter, if made in two rings, the upper ring may be  $\frac{3}{16}$  in., the lower ring and bottom  $\frac{1}{4}$  in., from 30 to 35 ft.  $\frac{1}{4}$  in. throughout, from 35 to 40 ft.  $\frac{1}{4}$  in. sides and  $\frac{5}{16}$  bottom.

For single riveted lap joints the following sizes are suitable :

For $\frac{1}{4}$ plates	$\frac{3}{8}$ in. rivets	1 $\frac{1}{8}$ in. pitch	1 $\frac{1}{8}$ in. lap
" $\frac{1}{2}$ "	$\frac{1}{2}$ "	1 $\frac{1}{2}$ "	1 $\frac{1}{2}$ "
" $\frac{3}{4}$ "	$\frac{5}{8}$ "	1 $\frac{3}{4}$ "	1 $\frac{3}{4}$ "

Angle iron rings at top and bottom vary from 2  $\frac{1}{2}$  in.  $\times$  2  $\frac{1}{2}$  in.  $\times$   $\frac{1}{4}$  in. for 20-foot vats, up to 3  $\frac{1}{2}$  in.  $\times$  3  $\frac{1}{2}$  in.  $\times$   $\frac{5}{8}$  in. for 40 feet diameter, and the top angle may be  $\frac{1}{2}$  in. narrower and  $\frac{1}{8}$  in. less in thickness than the bottom one.

When ordering vats for export ready for erection, the above particulars, in addition to the over-all dimensions and position of discharge doors and other attachments, are sufficient. With regard to limiting sizes of steel plates, the following information, kindly furnished by P. & W. Maclellan Ltd., of Glasgow, in answer to our enquiry, will be useful.

Limiting sizes, to be used without incurring extras, 25 feet by 5 feet for plates  $\frac{1}{4}$  in. thick, and 25 feet by 6 feet for plates  $\frac{5}{16}$  in. thick.

As regards the arrangement of plates, the same firm prefer to place the side plates vertically, each being the full depth of the tank, rather than in horizontal rings with lap joints.

When, however, tanks have to be built at or near the mines, there are two reasons why the latter arrangement is usually adopted. First, because only small plates can be obtained ; and secondly, because the vertical arrangement requires bending rolls, whose width is equal to the depth of the tank, and such heavy machinery is not usually available in mining districts. The best joints can be obtained, under these circumstances, by using butt joints with cover plates for the vertical seams and lap joints for the horizontal seams (see fig. 66). Also, in the bottom, the ends of the plates may with advantage be butted and covered. The only objection to this is the cost of cover plates and extra riveting.

In laying out the plan of vat bottoms with small plates, the position of

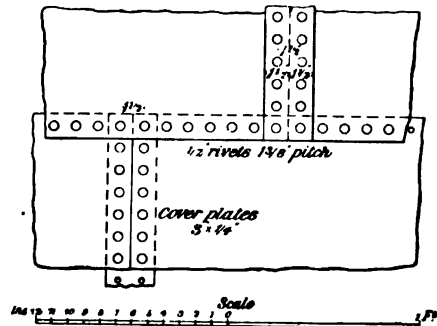


FIG. 66.—Tank Joints.

the discharge doors should be first determined, and the plates so arranged that the lines of rivets are clear of the doors. The reduction of waste to a minimum in cutting the circle should also be carefully considered. In fig. 67 a suitable arrangement of small plates for a vat 30 feet in diameter is illustrated. The projecting plates on the right-hand side of the figure are made of such lengths that the halves marked A, B, and C will be large enough to provide for the

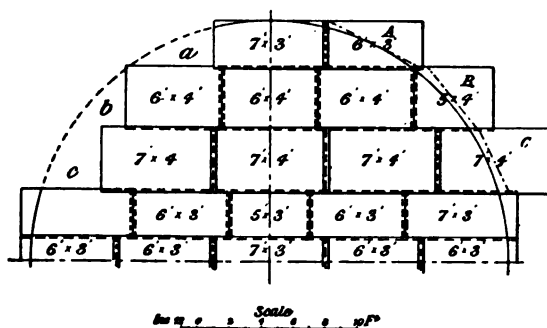


FIG. 67.—Arrangement of Bottom Plates.

spaces *a*, *b*, and *c* on the left-hand side, so that the quantity cut to waste is very small. But, although it is quite possible to make satisfactory tanks with small plates, it is clear that, from every point of view, except as to time of delivery, the advantage lies with tanks constructed of large plates in well equipped works.

When the plates are marked off for the rivet holes, they should also be numbered for erection, so that there shall be no difficulty in assembling them in their proper relative positions.

For erection, the following amount of labour is required in addition to a foreman—one riveter, one heater, and one holder-up, and two or three labourers. A gang of this number should drive from forty to fifty rivets per hour of actual riveting, the number in any particular case depending slightly upon the size of rivet, but chiefly upon the facilities provided for heating and supplying the rivets. In bolting the bottom plates together, preparatory to riveting, they must be packed up sufficiently to allow the holder-up to get underneath. When the riveting of the bottom plates and angle iron is completed, the whole bottom is lowered gradually and evenly on to its permanent seat. The side plates are then bolted to the angle ring and to each other temporarily, and the riveting of the side proceeded with. When the sides consist of two or more horizontal belts, each of these is finished separately.

With plates of  $\frac{1}{4}$  inch or more in thickness, the joints can then be caulked in the ordinary manner with a tool ground to a cutting angle of about  $70^\circ$ . With thin plates which cannot be caulked in a satisfactory way, the joints are sometimes tarred before riveting. Some engineers recommend an insertion of tarred canvas in the joints of thin plates, while others rely on the riveting only. Probably, the latter is the best practice, as any small leaks can be taken up after the tank is completed by filling it with water containing fine slime well stirred up, or by using saturated lime water. In either case, when the leaks have been stopped, it is a good plan to rub paraffin wax or vaseline well into the seams on the inside of the tank.

Fig. 68 illustrates one way of providing for the fixing of the filter cloth at its circumference in a steel tank. A light bulb iron *a* is riveted on to the inside of the bottom curb ring *b*, leaving a groove *c* for the edge of the cloth between the bulb iron and the side plate *d*. The rounded edge of the bulb does not cut the cloth, which may be held in the groove by wooden wedges. Where very light sections of bulb cannot be obtained, flat bar iron may be used, but the upper edges should be well rounded off.

**Steel Collecting-Vats.**—As with wooden vats, these are simply made by adding a peripheral overflow trough, and sometimes one or more slat gates. One method of constructing the trough is shown in fig. 69, which requires no explanation. The slat gates are usually of timber, and practically identical with the form shown in figs. 64 and 65.

**Quantities in Steel Vats.**—The sizes given on page 257 have been used in calculating the weights given in Table LVI. These weights are useful, not only for estimating the cost of the vats, but also in designing the supporting structure, especially when columns and girders are used.

TABLE LVI.—*Weight of Material in Steel Vats, including waste in cutting, also Angle Irons, Rivets, and Laps.*

Inside Depth in feet.	Diameter in feet.				
	20	25	30	35	40
6	6,390	9,020	15,180	19,720	29,410
7	6,920	9,660	16,300	20,990	30,780
8	7,440	10,290	17,370	22,240	32,160
9	7,960	10,910	18,380	23,470	33,550
10	8,470	11,550	19,390	24,690	34,950

The figures given for 30 ft. diam. appear, at first sight, to be too large, relatively to those in the other columns. This is accounted for, however, by the change from  $\frac{3}{16}$ - to  $\frac{1}{4}$ -in. plates at that diameter.

For collecting-vats, take a greater depth to allow for the metal in the trough and the extra rivets.

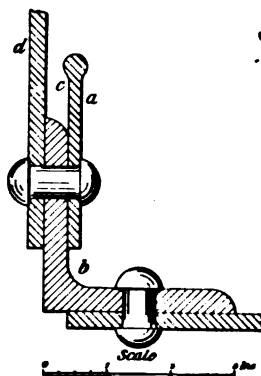


Fig. 68.—Fixture for Filter Cloth.

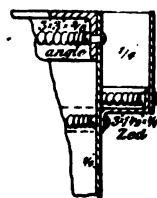


Fig. 69.—Trough for Collecting Vats.

**Masonry Tanks.**—In a very few cases, cyanide tanks have been formed by excavations lined with brickwork set in cement, and in very hard, solid ground these have proved satisfactory from an engineering point of view. But they should never be constructed on sites where there is any chance of settlement, for, although this may be sufficiently overcome as regards the strength and stability of the tank considered merely as a mechanical structure, it may, nevertheless, cause fine cracks in the cement lining, which result in leakage of solution that cannot be recovered, or even accurately gauged.

Brick tanks have also been constructed on the surface of the ground, but this method is also very seldom adopted at the present time, so that it is not necessary to give any detailed rules for their design. The walls must be designed in the same way as low masonry dams, but for a fluid weighing 120 lbs. instead of 62.4 lbs. With this difference kept in mind, the ordinary rules given in engineering books of reference will apply.

#### FOUNDATIONS AND SUPPORTS FOR VATS.

**Timber Framing.**—Vats should be carried on timber supports only in temporary plants, or in cases where timber is so cheap that the saving in first cost will outweigh the disadvantages of this construction. Fig. 70 shows a

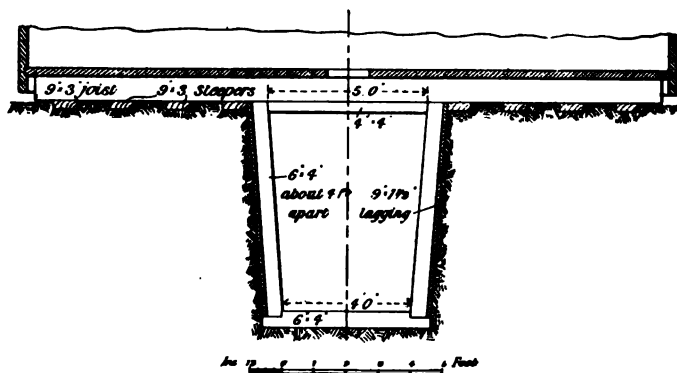
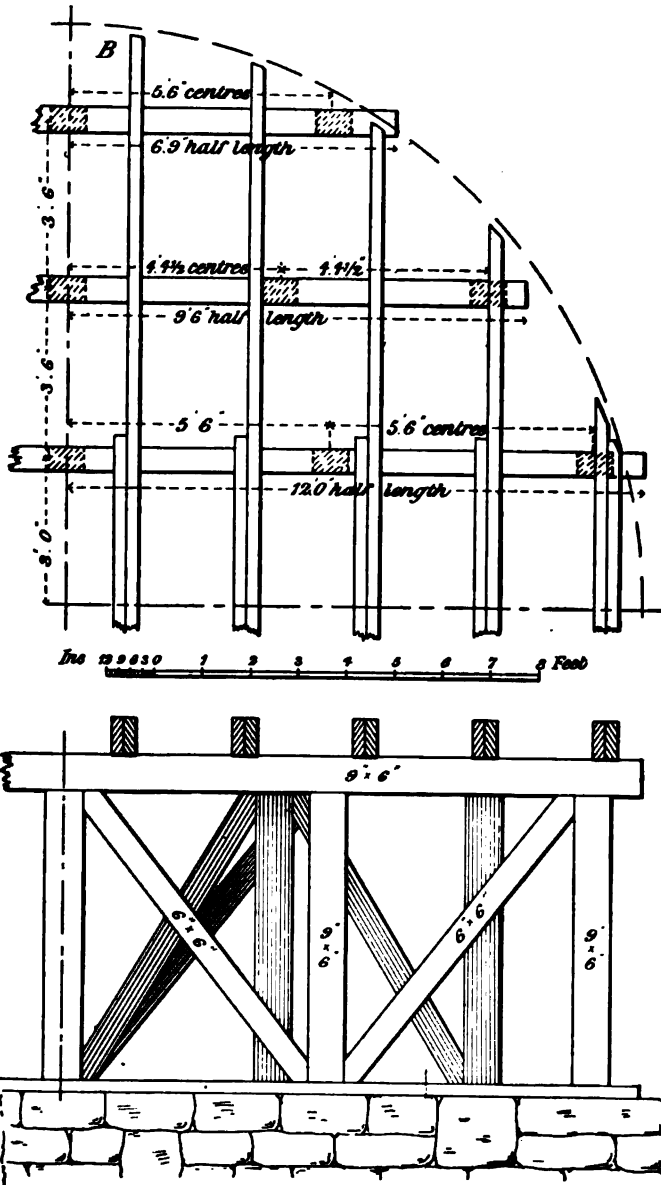


FIG. 70.—Timber Lined Tunnel.

timber lining for a tunnel excavated in the ground, and the cheapest method of supporting a leaching vat above it. This would be suitable for a temporary plant on sloping ground of the general design shown in figs. 34 and 35. The sleepers should be solidly bedded, well beaten down, and allowed to settle for a few days, under a considerable weight if possible, before the joists are permanently fixed upon them. Each joist should be then carefully fitted to the sleepers so that all the joists are flush and level. The framing, lagging, and sleepers should be well tarred before putting in place; and if the work is well done, this construction may be relied upon to last for several years. The

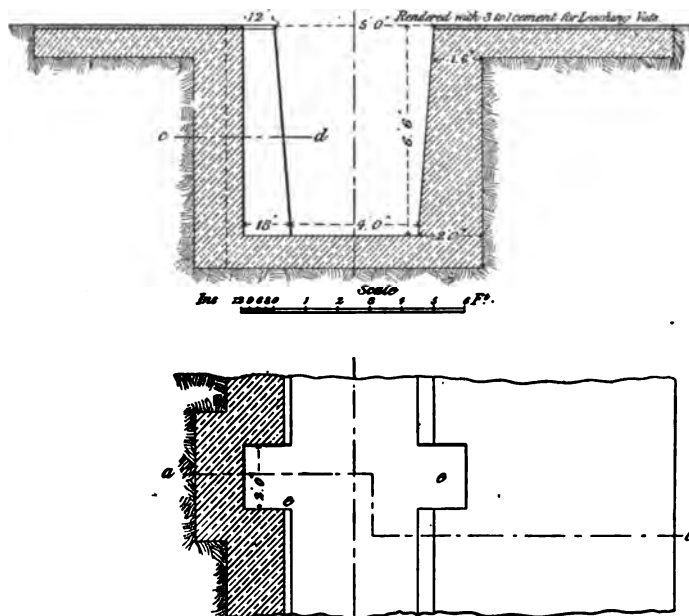
frames should be placed at such distances apart that each frame comes under a joist, to which the top distance piece can be nailed. Otherwise the latter would need to have a much greater scantling.



FIGS. 71, 72.—Timber Foundation Frame (Plan and Section).

Fig. 71 is a part plan of a timber frame for carrying a 24-ft. vat above the ground, and fig. 72 is a section through the centre of the same. In designing

a support of this kind, it is necessary to lay out the circle of the vat, and the position of the discharge doors and joists. The frames can then be arranged to leave a clear track under the doors, and also to support the joists to the best advantage. Thus, for instance, particular care should be taken that overhanging ends of joists, such as that shown at B, shall be as few and as short as possible. A sufficiency of diagonal struts in *both* directions, namely, in the separate frames and at right angles to them, is also of the utmost importance.



FIGS. 73, 74.—Masonry Lined Tunnel ; Section on *ab* and *cd* and Half Plan.

Framing of this description should never be bedded on the ground, but upon good stone sleeper walls built in lime or cement, or upon well driven piles.

The following table gives average quantities of timber in the two kinds of supports above described.

TABLE LVII.—Quantities in Timber Supports, exclusive of Joists.

Diameter of Vats in feet.	As in Fig. 70.		As in Figs. 71 and 72.	
	Square feet one inch thick.	Cub. feet.	Square feet one inch thick.	Cub. feet.
16	684	57	1389	116
18	850	71	1488	124
20	920	77	1646	137
22	1100	92	1836	153
24	1188	99	2161	180

**Masonry Foundations for Vats.**—These are almost invariably built in hammer-dressed rubble masonry, worked to level courses about every 2 feet of height, set in good hydraulic lime, and pointed with Portland cement on

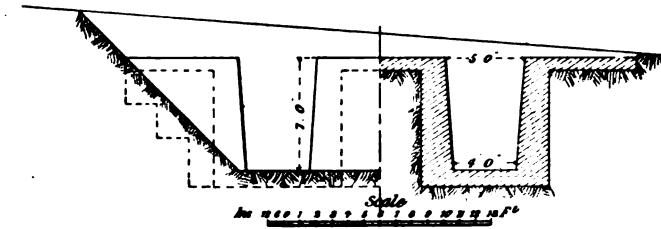
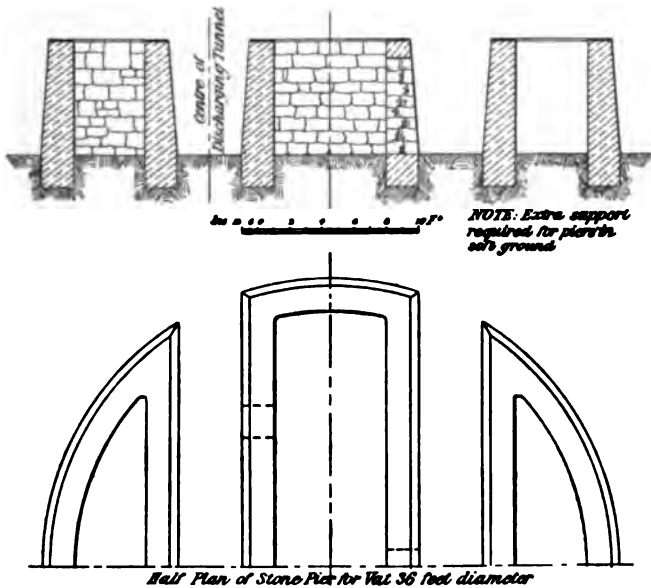


FIG. 75.—Half Elevation at End of Tunnel and Half Cross-section through Tunnel.

exposed faces. When the bottom of the vat is about level with the surface of the ground, so that the discharge tunnel has to be excavated, the type shown in figs. 73 and 74 will be found suitable. The minimum clear height in the tunnel should be 6 feet to allow men to work comfortably, and therefore efficiently. A fall of at least 1 in 100 should be given to the tunnel floor for



FIGS. 76, 77.—Pier for Single-tier Vats ; Section and Half Plan.

drainage ; and where trucks are used for discharging, refuge recesses, as at *e*, fig. 74, should be formed at intervals in the side walls. Where rope haulage is used, these recesses should be particularly large and frequent, but it is still better in this case to make the tunnel of greater width than shown in the



drawings, which represent the minimum required with 20 cubic feet side tipping trucks, worked by hand. Fig. 75 shows a suitable section for the foundation of a 32-foot vat, with two tunnels underneath. Foundations such

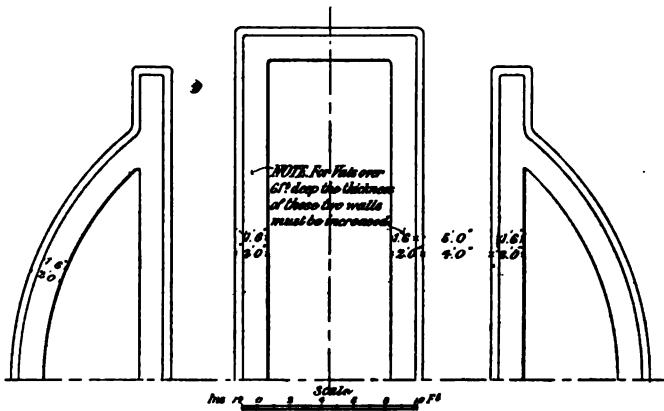


FIG. 78.—Pier with Projections for carrying Staging; Half Plan.

as these would be used in plants arranged as in figs. 34, 35 for the treatment vats, and also for the lower vats in figs. 36, 37.

When vats have to be supported above the surface of the ground, the nearest arrangement consists of a circular pier, with several inside parallel

walls to carry the vat joists, as shown in figs. 76 and 77. The stone work is only carried down 2 feet below the surface level in fig. 76, and this depth is also assumed in the quantities given in Table LVIII. on next page. But, of course, in each case the actual depth is determined by the nature of the ground as proved by trial excavations, and the importance of this point cannot be overestimated.

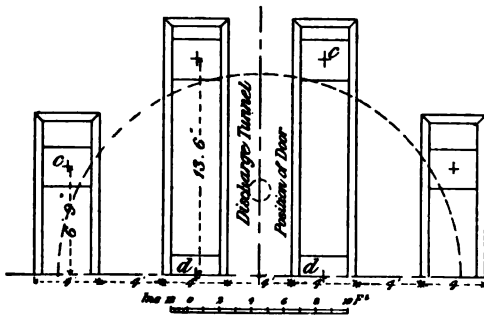


FIG. 79.—Pier for Double-tier Vats; Half Plan.

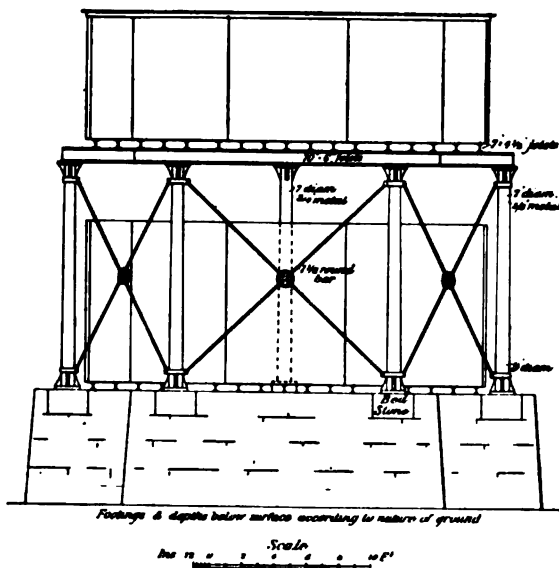
Sometimes circular piers are built with projecting portions, as in fig. 78, for carrying the trestles of overhead staging or the iron columns for an upper tier of vats. Another arrangement often adopted consists of a number of straight parallel walls, as in fig. 79, which is adapted to carry a lower vat 25 ft. diam. and eight cast iron columns supporting an upper vat. In this case properly dressed bedstones *cc* must be built into the walls to receive the bases of the columns. This is the ordinary form used in double-tier plants, such as that shown in figs. 41 and 42,

When necessary, manholes should be provided to allow access to the under side of the tanks, as in figs. 76 and 77, where two openings for this purpose are shown in plan and one in section at *aa*.

**TABLE LVIII.**—*Quantities in Masonry Foundation in cubic yards for one Vat, when built on Cemented Gravel or other hard soil, two feet below Floor Level.*

Diam. of Vat in feet.	As in Figs. 73, 74. Single Tunnel.	As in Fig. 75. Two Tunnels.	Circular Pier. Single Tunnel.	Circular Pier. As in Figs. 76, 77. Two Tunnels.	Parallel Walls, as in Figs. 78, 79, 80. Single Tunnels.	Parallel Walls, as in Figs. 79, 80. Two Tunnels.
20	36·41	...	46·10	...	...	...
24	46·75	69·50	60·64	...	108·76	...
28	59·21	85·75	88·15	88·15	131·32	165·28
32	71·99	101·90	...	101·82	...	220·37
36	...	120·74	...	116·74	...	238·00
40	...	142·50	...	161·48	...	260·37

Table LVIII. gives average quantities of masonry for several sizes of vats for some of the arrangements above described. These figures, together with the tables of prices in Chapter XLII., are useful in making provisional estimates.



**FIG. 80.—Pier for Double-tier Vats : Elevation.**

The blank spaces are left because they correspond to conditions which are unlikely to occur in practice. Some diameters lend themselves to the spacing of the walls better than others, hence the apparent discrepancies in the last

four columns. The figures are, however, correct, as they have been taken out from drawings made to a scale of  $\frac{1}{4}$  inch to a foot.

**Iron and Steel Supports for Vats.**—When steel vats are used, rolled steel joists are now always employed to support them, unless old rails or some other cheap substitute happens to be available. In single-tier plants, and also in those with two tiers when the lower vats are carried on masonry walls, only a single set of fairly light joists is required for each vat, as in fig. 80. But in other cases these light joists rest upon main girders, which in turn are carried upon columns, as shown in the upper part of fig. 80 and in fig. 81.

In many mining districts, land transport is a large item in the total cost of material of this description, and it is therefore the truest economy to use only the very best quality, such, for instance, as Siemens-Martin steel, in order

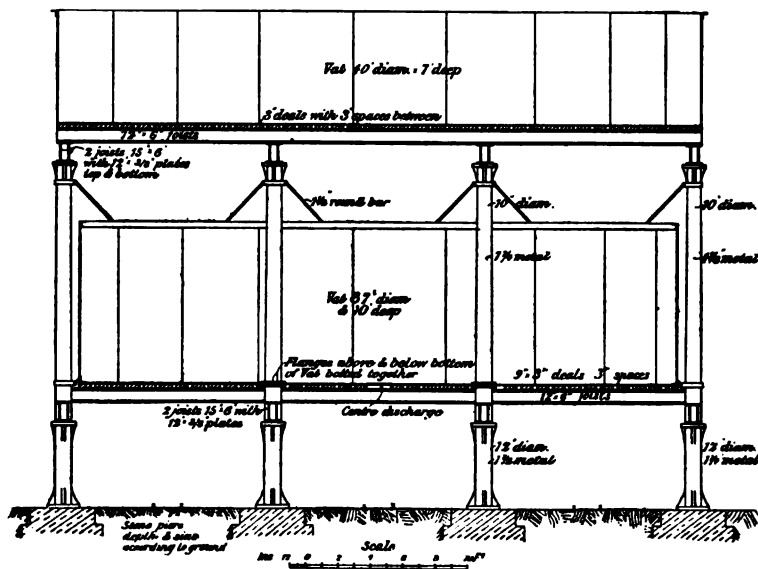


FIG. 81.—Pier of Iron Supports for Double-tier Vats ; Elevation.

to keep the weight of the material as low as possible. Most makers and dealers issue tables of stock sizes, with the corresponding weights and strengths, and in designing structures of this kind the best course is to obtain such tables for the particular brand which is to be used, and then to arrange the spacing of the joists and girders to suit some of the current market dimensions.

It is necessary, first of all, to ascertain the maximum distance allowable between the joists which immediately support the bottom plates of the vat, this distance, of course, depending upon the thickness of the plates, and the load resting upon them. An approximate result may be obtained by considering only a strip of plate, say 12 inches wide, and calculating as for a solid beam of this width whose depth is equal to the thickness of the plate. In doing

this we shall, for the sake of simplicity, neglect the weight of the plate itself, as this forms a very small fraction of the total load.

For a 1 vat foot in depth the load is 120 lbs. per square foot, and with a factor of safety = 4 the breaking weight must be 480 lbs. per square foot, or 40 lbs. per running inch of our hypothetical beam. Therefore if  $L$  = distance unsupported in inches and  $d$  = depth of vat in feet, the total breaking weight must be  $40 dL$ , and the ordinary formula becomes—

$$40dL = \frac{12KB\ell^2}{L}$$

Whence

$$L^3 = \ell^2 \times \frac{12KB}{40d}$$

The breadth  $B$  is by assumption 12 inches, and for steel plates  $K = 12,000$  lbs.

Therefore

$$L = \frac{208\ell}{\sqrt{d}}$$

For a depth of 10 feet, which is rarely exceeded nowadays for sand vats, this formula gives the distances shown below ; and allowing usual widths for the joists, we have also given the dimensions between centres.

For $\frac{1}{4}$ inch plates $L = 12.3$ inches.				Between centres say 15 inches.			
„	$\frac{1}{4}$	„	„	16.5	„	„	20
„	$\frac{1}{2}$	„	„	20.6	„	„	24
„	$\frac{3}{4}$	„	„	24.7	„	„	30

Vats have been erected in which the above figures are exceeded by as much as 25 per cent., and the plates have stood the strain, but in these cases the factor of safety must be less than is desirable for permanent structures. In some cases a floor of 3-inch deals laid on the flat about 3 inches apart is interposed between the vat and the joists. When this is done the latter may be placed at 4-foot centres for a vat 10 feet deep. The question whether it pays to use the timber in this way may be settled, under any given conditions as to relative cost of wood and iron, by finding the weight of joists required in either case, in the following manner.

For  $\frac{3}{4}$ -inch plates without timber and with joists at 15-inch centres, the load per joist on an 8-foot space is 5.4 tons, and a 6-in.  $\times$  2 $\frac{1}{2}$ -in. joist weighing 14 lbs. per running foot is sufficient. So in this case the weight of steel per square foot of vat supported thereby is 11.2 lbs.

Similarly we obtain the following figures :

For $\frac{1}{4}$ -inch plates, joists weigh 10.2 lbs. per sq. ft. of vat.			
„	$\frac{1}{4}$	„	10

With the deal floor and joists at 4-foot centres, the weight of joists is 8 lbs. per square foot of vat. So the saving at the best is only 3.2 lbs. of steel against the price of 1 running foot of 9-in.  $\times$  3-in. deal. At the usual relative

\* Derived from formula for solid beams fixed at both ends,—see Molesworth, 23rd edition, p. 136.

prices of the two materials, it is clear that there is little difference in cost either way. Probably with  $\frac{3}{16}$ -inch plates the deals cause a slight economy, but with thicker plates it is cheaper to omit them. Their only advantage is that they afford a more yielding bed for the vat to rest upon.

In laying out the main girders, it is necessary to keep in mind the well known fact, that the total weight of metal in the girders for any given load increases with the span, so that if the columns are too far apart the weight both of main girders and of joists will be unduly great. The Table LIX., when the cost per ton of the material is known, enables the designer to determine very quickly the extra cost of any proposed increase of span, or *vice versa*. The light sections are applicable to the joists and the heavy sections to the main girders.

TABLE LIX.—*Showing relation between Weight of Rolled Joists of Siemens-Martin Steel and the Safe Load carried by them when Factor of Safety = 4. First column of each pair relates to a long ton of 2240 lbs. The second is obviously applicable to short tons or any other measurement.*

Span in feet.	Light Sections from 4 in. x 3 in. to 10 in. x 6 in.		Heavy Sections from 12 in. x 5 in. to 18 in. x 7 in.	
	Weight of Joists in lbs. per ton of load.	Weight of Joists expressed as per- centage of load.	Weight of Joists in lbs. per ton of load.	Weight of Joists expressed as per- centage of load.
6	14.96	0.67	...	...
8	23.72	1.05	...	...
10	30.37	1.36	...	...
12	39.79	1.77	21.07	0.94
14	49.20	2.20	27.41	1.22
16	62.72	2.80	35.81	1.60
18	74.96	3.35	45.48	2.03
20	87.25	3.89	55.95	2.50
22	...	...	68.25	3.05
24	...	...	81.29	3.63
26	...	...	95.84	4.28

**Columns.**—Cast iron is generally used, but in cases where transport to the mine is exceptionally difficult, stanchions of rolled steel sections may be recommended as being less liable to accidental breakage. As shown in Table LX., there is very little difference in the weight of material required in either case, but cast iron is cheaper per ton. From this table the quantities required for any given load can be readily obtained for estimating purposes.

Fig. 81 shows a pair of superposed vats, both of which are carried upon cast iron columns.

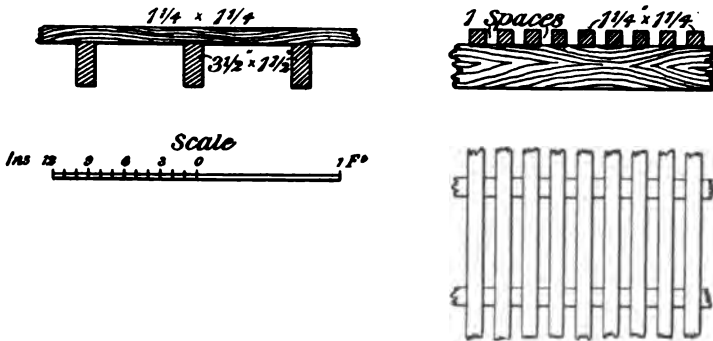
In fig. 80 the masonry walls are the same as those shown in plan in fig. 79, and the upper and lower vats are each 25 feet in diameter. It is, however, also quite common, and indeed somewhat better practice when all the vats are

in one line, to have the top vat of larger diameter and less depth than the lower one, so that part of its weight comes directly over the columns, as shown in fig. 81, thus leaving less weight to be supported by the girders. The columns should be stayed by diagonal ties of  $1\frac{1}{8}$  to  $1\frac{1}{2}$  inch round iron

TABLE LX.—*Relation between Weight of Columns and Loads sustained by them.*

Height of Column in feet.	Cast iron Columna. Factor of Safety = 5.		Steel Columna. Factor of Safety = 4.	
	Weight of Column in lbs. per ton of load.	Weight of Column as percentage of load.	Weight of Column in lbs. per ton of load.	Weight of Column as percentage of load.
6	4.56	0.20	4.95	0.22
8	6.02	0.27	6.45	0.29
10	7.96	0.36	8.32	0.37
12	10.22	0.46	10.76	0.48
14	13.14	0.59	14.11	0.63
16	16.67	0.74	18.29	0.83

between them, and means must be provided for tightening these up when in place, such as the central ring with nuts inside it, shown in the drawings. Holding-down bolts for the columns must be built in the masonry, and the main girders should be connected by plates and bolts where they meet on top



FIGS. 82, 83, 84.—Filter Frames.

of the columns, and the joists also bolted to the girders where they cross the latter over or close to a column, thus binding the whole of the separate parts into one complete and rigid structure.

**Filter Frames.**—The filter cloths of leaching vats are supported on wooden frames, which are generally constructed in the following manner. A number of slats laid on edge are parallel to each other, at about 6 to 9 inch

centres, and on the top of these, at right angles to each, there are nailed a series of wooden strips, with 1-inch spaces between them, as shown in figs. 82, 83, and 84.

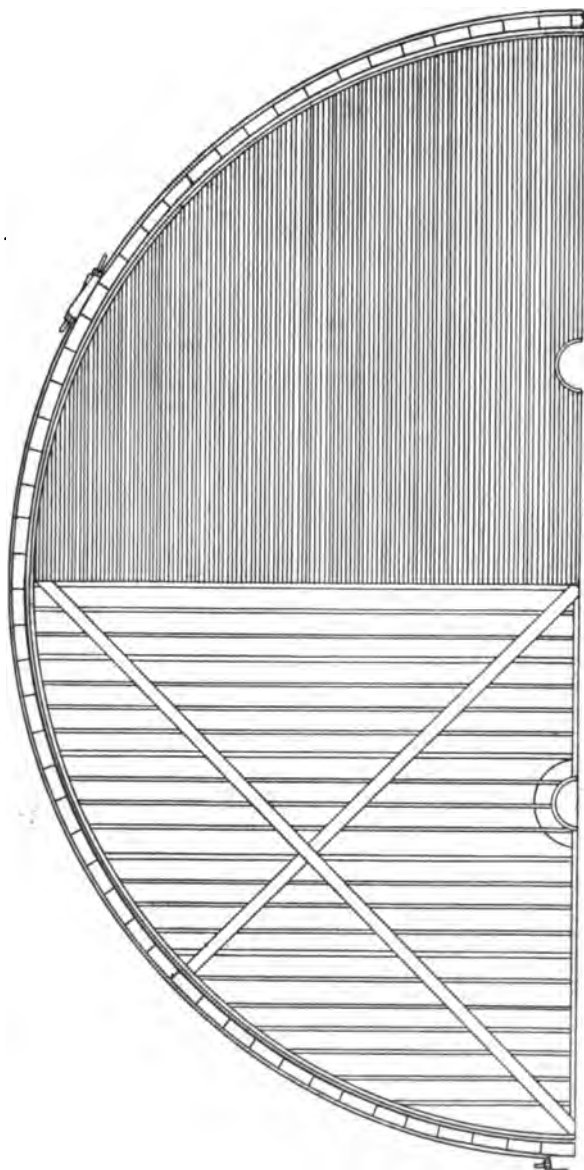


FIG. 85.—Filter Frame.

For wooden and masonry vats the usual dimensions are:—Slats 3 inches  $\times$  1 in. at 6-in. centres. Strips 1 in.  $\times$  1 in. The quantity of timber in such a frame is exactly equal to the area covered by 1 in. thick, or is equal to 0.0833 cubic ft. for each square foot of vat area.

For large steel vats the sizes shown in figs. 82 and 84 are suitable. In this case the quantity of timber per square foot of vat is 1.272 square feet, reduced to 1 in. thick, or 0.106 cubic foot. The frame should be made in several separate sections, each section being braced by diagonals let in flush with the tops of the slats before the strips are nailed on, as shown on the left-hand side of fig. 85. The ends of the slats and strips should be about 1 inch clear of the sides of the vat.

The filter cloth lies on the strips, and in wooden vats is sometimes nailed to the sides by  $\frac{3}{4}$ -inch clout nails. Otherwise a curved strip of wood or iron (see fig. 68) is fixed round the inside of the vat, and the edge of the cloth is tucked in and wedged between this and the side of the vat. Sometimes boards pierced with numerous holes are laid flat on the slats instead of the strips to support the cloth, but this is inferior to the method illustrated, because it considerably reduces the effective filtering area.

**Filter Cloths for Vats.**—Cocoa-nut fibre matting is the most usual material for this purpose, and it may be used alone, or preferably with a cloth of hessian above it. The matting should be of the best quality obtainable, but as regards the hessian, a medium quality should be selected, as it filters better than the more expensive kinds. Canvas and duck have been occasionally tried, but cannot be recommended for this purpose.

The filter cloths are made up to a circle 10 to 12 inches larger than the vat, and when made in sections, these separate parts should overlap at least 6 inches when laid in place, this overlap being in addition to the extra diameter above mentioned. The outside edges are bound with strong canvas 6 inches wide, and the edges round discharge holes with 4-inch canvas. Instead of making the holes in the main filter cloth a tight fit round the discharge tubes, it is more convenient to use at each door a separate hessian mat, about 3 ft. 6 ins. square, with a central hole in it, so cut and bound that it will spring tightly around the discharge tube, and so cover the larger hole in the main cloth.

**Shovelling Strips.**—To preserve the filter cloth from damage while shovelling out the residues, strips are lightly fixed above it, at right angles to the strips below. Pieces 2 in.  $\times$  1 in. laid flat are sufficient for the purpose, with 6-inch spaces between them, and they should be fixed with as few screws as possible, so as to be readily removed.

**Discharge Doors for Vats.**—Bottom discharge doors, attached to the lower end of tubes passing through the filter cloth and frame, were introduced by Chas. Butters in 1891 in Johannesburg, and these are now used, sometimes with modifications in detail of construction, in all parts of the world. The side discharge door, whose lower edge is flush with the filter cloth, was introduced by W. R. Feldtmann about the same time, but, for the reasons given below, the latter is not as good as the former. For instance, in a 20-foot vat with one central bottom discharge, none of the material has to be shovelled more than 10 feet, and the average distance is 6 feet 8 inches, while, with a single side discharge in the same vat, some of the material has to be moved 20 feet, and



even with two side discharges, diametrically opposite to each other, the maximum distance is still 14 feet 2 inches. But the diagrams figs. 86 to 89 show more clearly than a written explanation the relative merits of side and bottom discharge.

Fig. 86 shows a vat with one central bottom discharge *a* and two side discharges *b b*. Fig. 87 has two of each kind. In each of these two diagrams

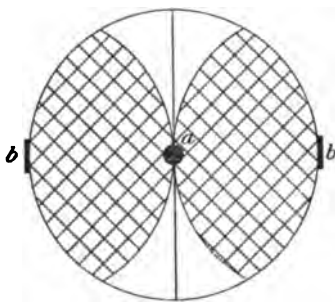


FIG. 86.—Discharging Residues.

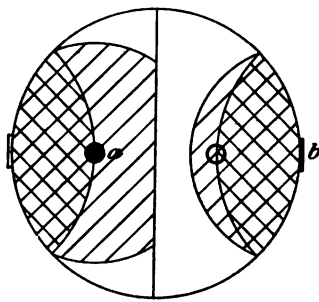


FIG. 87.—Discharging Residues.

the cross hatching indicates the portions of the vat contents which can be removed with equal labour by either system, that is to say, the portions which lie symmetrically between the rival doors. Further, in diagram fig. 87 the single-hatched portions show the additional amounts that can be discharged with the same maximum throw as for the cross-hatched portion; the left-hand side of the figure shows the extra amount for the bottom door, and the right-hand of the figure for the side discharge door. The advantage of the first-

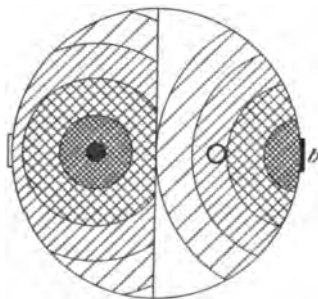


FIG. 88.—Discharging Residues.

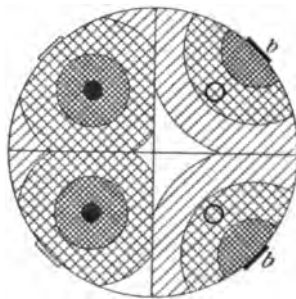


FIG. 89.—Discharging Residues.

named is very apparent; and it is also evident that in both figures the bottom discharge door is much more favourably situated for the removal of the remaining unhatched portions.

Fig. 88 shows on the left hand for bottom doors, and on the right hand for side doors, the amount of material that can be discharged by the two methods respectively, for the series of shovelling limits indicated by the various kinds

of hatching. Fig. 89 is a similar diagram for four doors of each kind. In both these figures the darkest shading shows the material included within a radius of one-eighth of the vat diameter, the next zone has a radius equal to one quarter of the diameter of the vat, and so on. The general result indicated is that, with a short throw, the bottom doors discharge twice as much as the side doors, and are therefore much more economical in practical work.

**Best position for Discharge Doors.**—Although it may appear to some to be an unnecessary and unpractical refinement, it is, on the contrary, distinctly

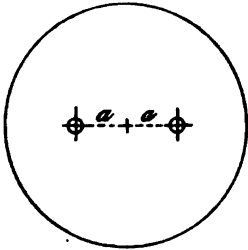


FIG. 90.—Vat with Two Discharge Doors.

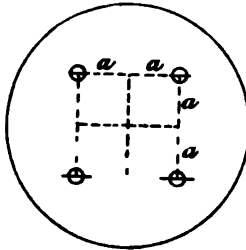


FIG. 91.—Vat with Four Doors.

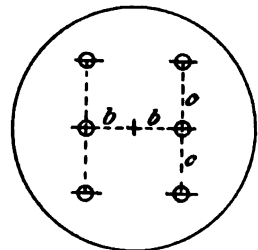


FIG. 92.—Vat with Six Doors.

advantageous in practice to take some trouble to ascertain the best possible position for the doors, because this affects the cost of removal of every tankful of residues. It is generally convenient to have the same total quantity of material delivered from each door, therefore the first rule is to divide the bottom of the vat into as many equal areas as the number of doors. The second rule is to place each door at the centre of gravity of its own section,

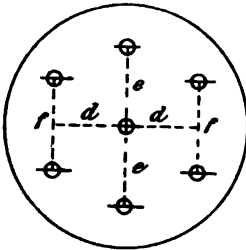


FIG. 93.—Vat with Seven Doors.

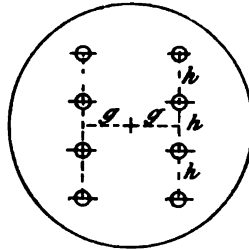
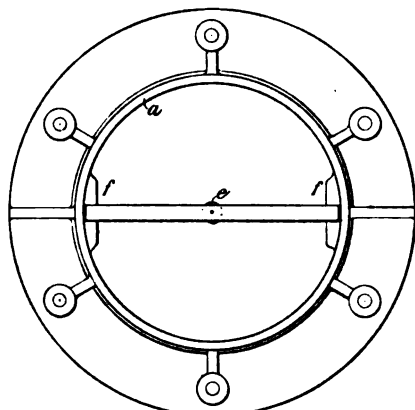
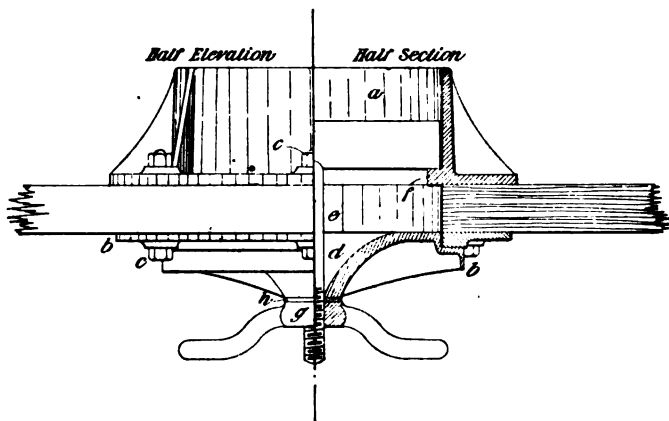
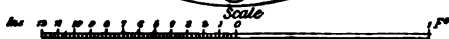
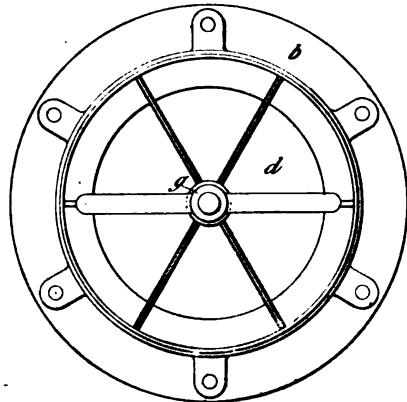


FIG. 94.—Vat with Eight Doors.

which can be easily found by the experimental method given in Molesworth's Pocket-Book (23rd edition, p. 367). Having done this, the positions have then to be altered sufficiently to bring the doors into a series of straight lines. This will then be the best result from theoretical and practical considerations combined, and the supporting structure of the vats should be laid out so that the discharging tram lines or conveyors can be placed along the lines so found. In figs. 90 to 94 we have shown suitable arrangements for various numbers of doors, which have been worked out on the lines above indicated.

*Plan from above**Plan from below*

FIGS. 95, 96, 97.—Butters' Discharge Door.

For two doors (fig. 90), the distance  $a$  = radius of vat  $\times .4244$ .

For four doors (fig. 91),  $a$  also = rad.  $\times .4244$ .

For six doors (fig. 92),  $b$  = rad.  $\times .4$  and  $c$  = rad.  $\times .55$ .

For seven doors (fig. 93),  $d$  = rad.  $\times .60$ ,  $e$  = rad.  $\times .66$ , and  $f$  = rad.  $\times .75$ .

For eight doors (fig. 94),  $g$  = rad.  $\times .4$  and  $h$  = rad.  $\times 4.13$ .

**The Butters Discharge Door.**—Figs. 95, 96, 97 illustrate a 16-inch door of cast iron applied to a wooden vat. A circular hole, 16 inches in diameter, is cut in the floor of the vat; a cast flanged tube  $a$  is placed above this, a flange  $b$  below the opening, and the two are bolted together by six  $\frac{3}{4}$ -inch bolts ( $c$ ) passing through the wood. The upper side of the door  $d$  and the lower side of the flange  $b$  are faced in the lathe to ensure a well-fitting joint. A tee bolt  $e$  rests on snugs  $f, f'$  which are cast in the tube  $a$ . This bolt passes through a central hole in the door, and the latter is screwed up tightly by a butterfly nut  $g$  on the tee bolt. A washer  $h$  is usually placed between the door and nut, and the bosses of both should be turned truly square with the axis of the bolt. When the door is screwed home, the tube is filled with sand and a slab of well worked clay is pressed carefully over the top of the tube  $a$  to make a water-tight joint. The tank is then ready for filling. When the residues are to be discharged, a truck is placed underneath and the door is taken off. Often-times it is sufficient to push a rod down through the sand to start it running through the door, but sometimes it is necessary to dig a hole down to the discharge tube. As it is somewhat dangerous to dig down more than 8 or 9 feet, an additional device has been used for vats of this depth and over, and this is shown in fig. 98. A sheet of  $\frac{1}{8}$  steel is bent to form a nearly complete tube  $i$ , leaving, however, a longitudinal slot  $k$  about 2 inches wide.

It is bound and stiffened by two rings of flat bar iron  $j, j$ , 2 in.  $\times \frac{1}{2}$  in. The lower end of the tube fits into the tube  $a$  of the discharge door, and if necessary the lower end of a second tube fits the top of the first, and so on. These tubes are fixed in place before the vat is filled, and consequently, for discharging, it is only necessary to dig down in the first place till the uppermost tube is reached. These tubes are successively removed as the work of discharging proceeds.

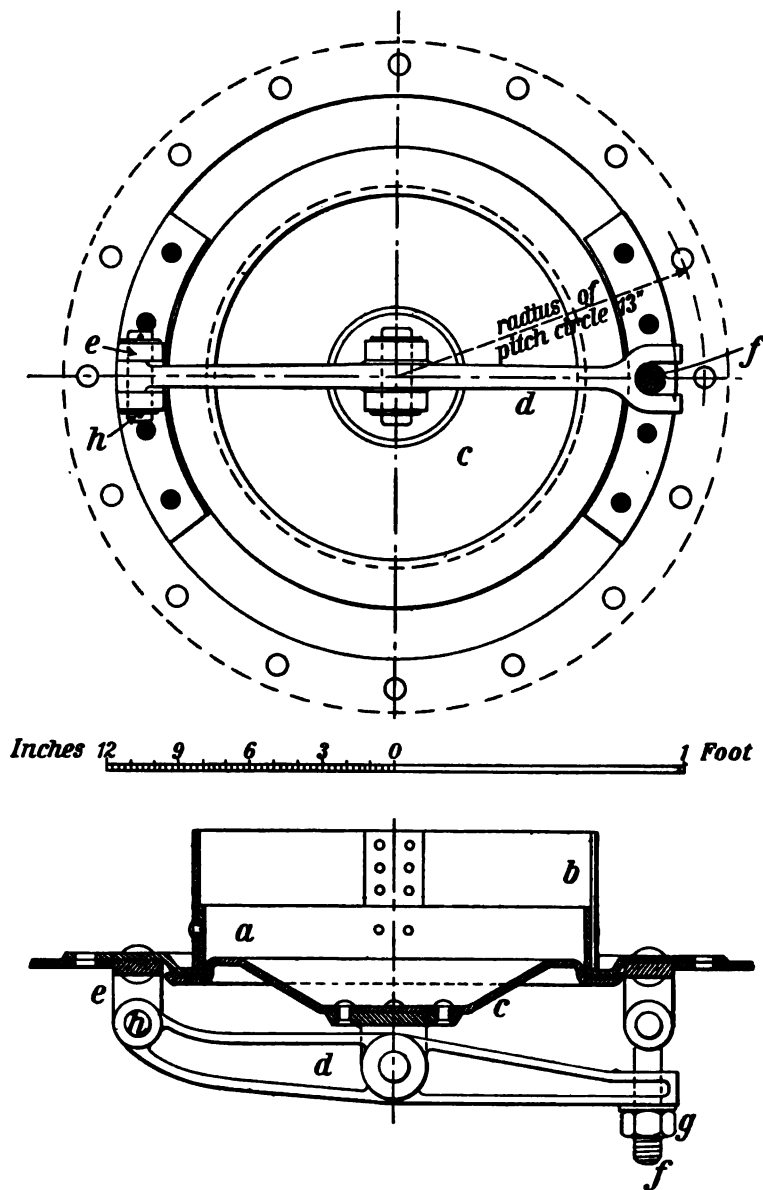
A quick way of piercing a hole and starting the sand to run out is by means of a fine jet of water under pressure, applied at the top directly over the discharge door. A half-inch hose pipe will pierce 10 feet in a few minutes by this means if a fairly high pressure is available.

A very simple form of door has been used in S. Africa and America, which consists merely of a taper plug and a plate with a hole of corresponding taper



FIG. 98.—Discharging Tube.

attached to the vat. A long handle fixed to the plug projects through the top of the tailings, so that the plug may be withdrawn when required. It is



FIGS. 99, 100.—Hinged Discharge Door ; Section and Plan.

not so safe, as regards leakage, as the more usual forms, and it sometimes happens that the plug sticks, and is not readily removed when required.

**Hinged Doors.**—Figs. 99 and 100 illustrate a very convenient form of discharge door made by the United Engineering Coy. of Johannesburg, to whose courtesy we are indebted for the drawings. It is made almost entirely of pressed steel, and consists essentially of a flanged tube made in two parts *a* and *b*, a door *c*, and a lever *d*, which is pivoted at its centre to the door, hinged to the bracket *e* at one end, and supported at its other end by a swinging bolt *f*. The details of its construction can be well understood from the figures without further description. For discharging, the nut *g* is slackened, the bolt *f* is swung to the right and the lever and door are swung to the left on the pin *h*. The door may be left hanging downwards, or it may be hooked back on to the bottom of the vat, out of the way of the workmen in the tunnel. It may be noted that the leverage obtained when screwing up is twice as great as it is with the central bolt in fig. 96, and by dispensing with the bolt hole in the door one possible source of leakage is avoided.

With either form of door a gasket of rubber or of asbestos cord can be used to make good the joint between door and flange.

Square doors hinged on one side are made by the Pacific Tank Coy. of San Francisco. In these a removable bridge bar is attached at both ends by bolts to the door frame, and carries a central set screw with a wheel handle to press the door home.

**Distributors for Collecting-Vats.**—The best automatic apparatus for delivering battery pulp to collecting-vats is the distributor devised by Chas. Butters and the late G. A. Mein, which is illustrated in figs. 101 and 103, these being a section and plan respectively. Fig. 102 shows the method of staying the central pipe which carries the apparatus.

The movable part consists of a casting *a*, carrying a sheet iron hopper *b*. This casting is keyed to a vertical steel shaft *c*, and is provided with six radial pipes *d d*, attached to it in the following manner. Short lengths *d<sub>1</sub>* of these pipes, having both ends screwed, are laid in the mould in the foundry and cast in, thus avoiding the cost of boring and tapping the holes in the casting *a*. A flat nozzle *e*, formed of sheet iron and brazed on to a pipe socket, is attached by a bend and a suitable length of ordinary black pipe and another socket to each of the short pipes *d<sub>1</sub>* in the casting *a*. The latter is bored out for the spindle and also to fit the lower casting *f* at the part marked *g*. The lower casting *f* is bored to form a bearing for the shaft *c*, and also turned to fit the upper casting *a*, the idea being to exclude all grit from the bearing. At its lower end a steel plug *h* is screwed in to serve as a footstep for the shaft. The plug is turned slightly convex on its upper end, and one or more turned steel washers *j* are placed between the shaft and plug. The plug and washers should each be hardened and tempered.

A pipe *k* with a flange at each end carries the whole apparatus, and is stayed by three rods *l l l* clamped on to it, as shown at *m* in fig. 102. The outer ends of these stay rods pass through the staves as seen at *n*, and are held by nuts and washers on either side of the timber.

An oil pipe *o* is provided for lubricating the bearing and footstep.

When the pulp enters the hopper *b* it passes out by the radial pipes *d d* and the nozzles *e*, and the reaction of the issuing water against the bends at the outer ends of the pipes *d d* is sufficiently powerful to cause these pipes to revolve in the direction shown by the arrows. In consequence of this rotation, each pipe delivers a stream of pulp over an annular space in the vat, and the pipes must be so adjusted, as to their relative lengths and diameters,

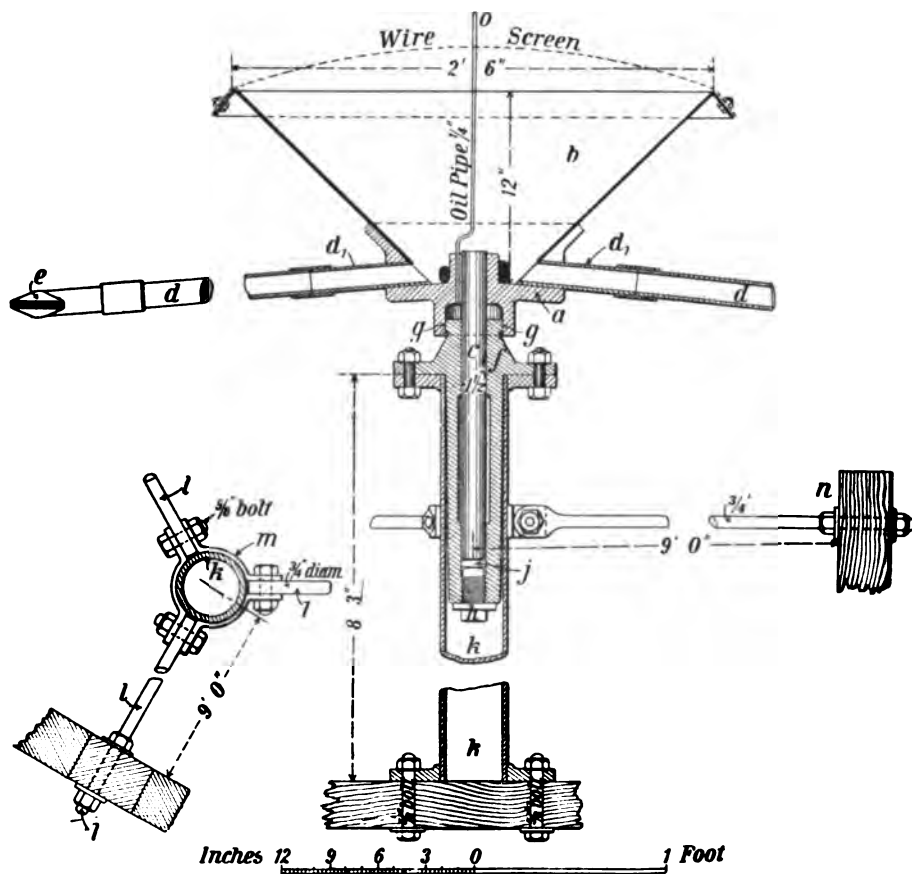


FIG. 101.—Butters-Mein Automatic Distributor in sectional elevation.

FIG. 102.—Plan of connection with staves.

that the supply of pulp is as nearly as possible equal on every unit of surface of the vat. For instance, if the pipes are of equal diameter, and the inclination so arranged as to give an equal discharge from each, then the area of the vat should be divided by circles into twice as many equal areas as there are pipes. The length of each pipe is then determined by the fact that it feeds two of the areas so obtained, and that, therefore, its nozzle must be over the circle which divides these two areas. If the radial pipes have different diameters, the annular spaces must be proportioned accordingly, and the most

uniform results are obtained by having the longer pipes of larger diameter than the short ones. Also, it is generally advisable to make the longest pipe less than the length determined as above, so that it shall not deliver too close to the outer edge of the vat; and to achieve this practical requirement, the best plan is to design the pipes strictly according to the principles laid down,

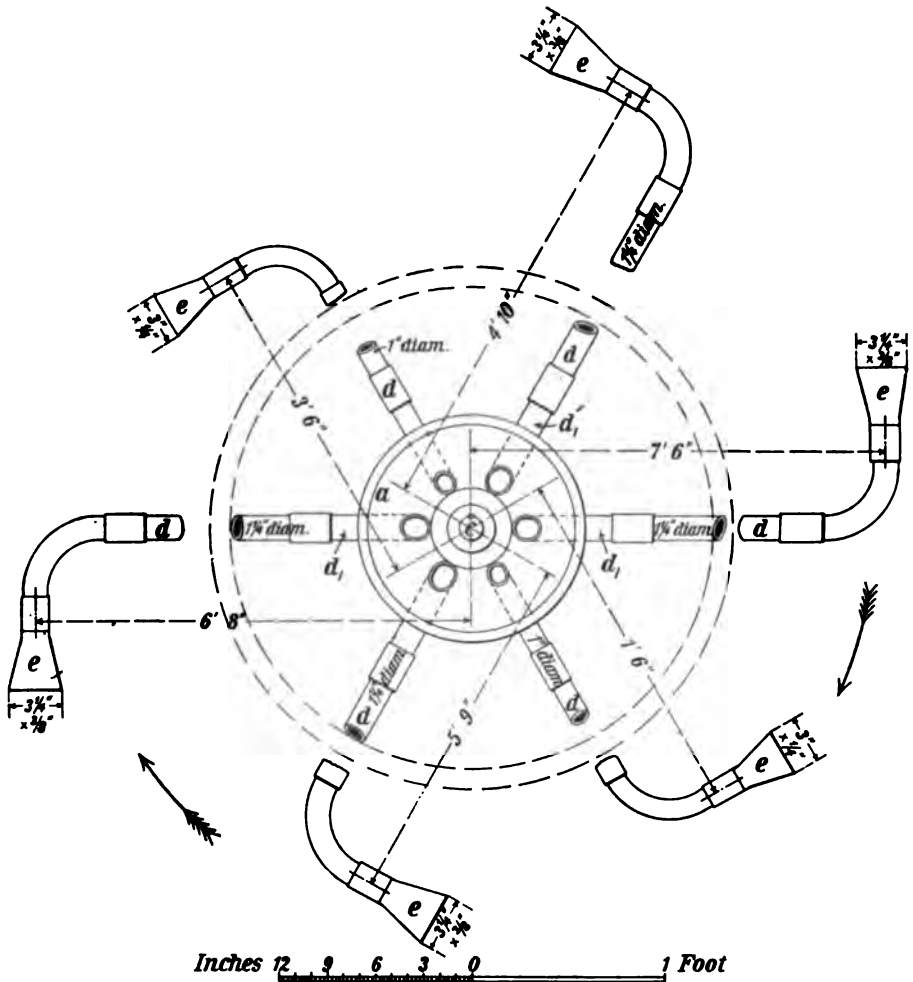


FIG. 103.—Butters-Mein Automatic Distributor in Plan.

but for a vat 1 foot less in diameter than that in which they are to be actually used.

A wire screen with about three holes to the linear inch should be attached to the flange of the hopper by six small bolts and nuts, to prevent choking of the radial pipes by large pieces of ore or the like.



**Stirring Gear for Slime Vats.**—The sketches figs. 104 and 105 give a general idea of the arrangement of stirring gear generally used at present on the Rand. A timber

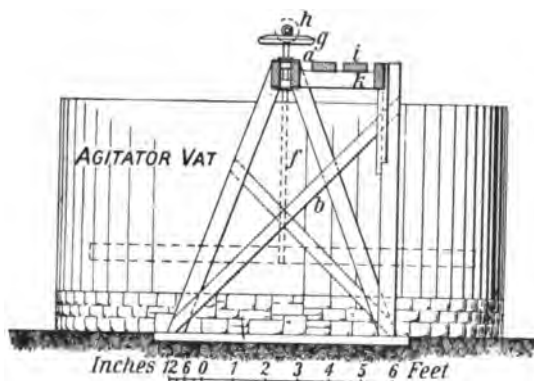


FIG. 104.—Stirring Gear: end Elevation.

platform *a*, supported on trestle frames *b*, is fixed over the line of vats, and carries a series of pedestals *cc*, with bearings for a horizontal countershaft *d*. The platform also supports vertical bearings *ee* for the paddle shafts *ff*. The latter are sometimes hung on these bearings by collars, and in other cases are provided at their lower ends with footstep

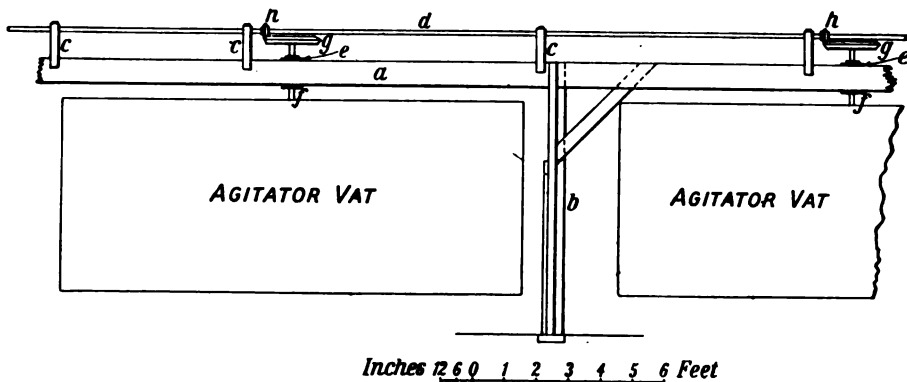


FIG. 105.—Stirring Gear: longitudinal Elevation.

In Western Australia, agitation vats are often arranged in the same way as the well known Wheeler pan, *i.e.* the bevel gearing is below the vat, and the paddle shaft passes up through a central tubular shield which forms an integral part of the vat. The paddles are attached near the top of the shaft, and pass down outside the central tube into the pulp.

## CHAPTER XXXVI.

### CONSTRUCTION OF OTHER ESSENTIAL PARTS OF A CYANIDE PLANT.

HAVING described the principal points of vat design and construction, the remaining features which are common to all plants may be dealt with in the following order:—

Precipitation appliances.

Apparatus for motive power.

Pumps.

Buildings to contain the above.

Piping and cocks.

**Zinc Boxes.**—Data taken from practice for determining the sizes of these, according to the quantity of solution to be dealt with, have been given in Chapter XX., so that it is only necessary here to describe the practical construction of the boxes. The box shown in figs. 106, 107, 108 is a fair specimen of the type generally used on the Witwatersrand, and of a size which is rarely exceeded. The sides, bottom, and ends are of  $1\frac{1}{2}$ -inch material, and this thickness should be used for all smaller sizes as well, because thinner stuff is liable to warp, and so cause leakage. Where timber is cheap or solutions rich, 2-inch planks may be used with advantage for the size illustrated. At the top and bottom of each side is a piece *a*, 3 inches square, to stiffen it, and to receive the vertical bolts *b* along the sides and the cross bolts *c* under the bottom. The upright stiffeners *d* are also 3 inches square, and so spaced that the horizontal bolts *e* through the box pass centrally between each pair of partitions. In cutting the notches *f* in these uprights, care should be taken that the part between the horizontal stiffeners *a* is sufficiently short to allow the vertical bolts to draw the side planks tightly together. The same remark applies to the notches at the ends of the  $4\frac{1}{2}$ -inch by 3-inch cross stiffeners *g* under the bottom. All the timber should be machine-dressed all over, and hand-dressed in addition on abutting edges, which are also grooved and tongued. The tongues are separate, except those on the lower edges of the sides and the outside edges of the bottom, where in each case the tongue is formed on the outside edge of the plank itself, so as to weaken the longitudinal stiffeners *a* as little as possible.

The partitions *h* may be of 1-inch clear pine, the widest obtainable being used so as to have few joints. They must be cut very accurately to the width of the box, in order that they may not prevent the tightening up of the

bottom planks, on the one hand, or leave spaces between themselves and the sides, on the other hand. The joints in the partitions are generally tongued and grooved, and the joints between partitions and sides are made water-tight by nailing on  $1\frac{1}{2}$ -inch triangular fillets *h*, as shown in plan in fig. 107.

The box is bolted together with  $\frac{1}{2}$ -inch bolts throughout, but instead of using ordinary  $\frac{1}{2}$ -inch washers, it is better to use plates about  $1\frac{3}{4}$  inch square by  $\frac{3}{16}$  thick under the nuts and heads, so as to have a good bearing surface on the timber.

Each compartment has a tray *i*, supported on fillets *k*, nailed to the sides of the box. One such tray is illustrated in figs. 109, 110. It has sides and ends 3 in. deep by 1 in. thick, screwed to triangular corner blocks *l*, and the rectangular frame thus formed is covered below with  $\frac{1}{4}$ -inch screening *m*. Handles *n* of  $\frac{1}{2}$ -inch round iron are attached.

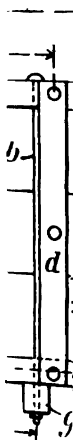
Sometimes zinc boxes are made with a trough along one side. In this case each compartment has a hole in the side fitted with a wooden plug, so that when these plugs are withdrawn the precipitated gold sludge can be washed out of the compartments along the trough into a clean-up vat. It is a question whether the saving of labour by this means is worth the increased risk of losing gold.

**Metal Zinc Boxes.**—Boxes made of sheet iron or steel can now be obtained from several makers in England and America, and are either enamelled or painted on the inside. In ordering these it is necessary to specify that the paint or enamel shall not only be proof against cyanide, but also against acid or alkaline solutions. For if the steel is exposed and comes into contact with the zinc, a couple is formed which wastes zinc and deposits gold on the box.

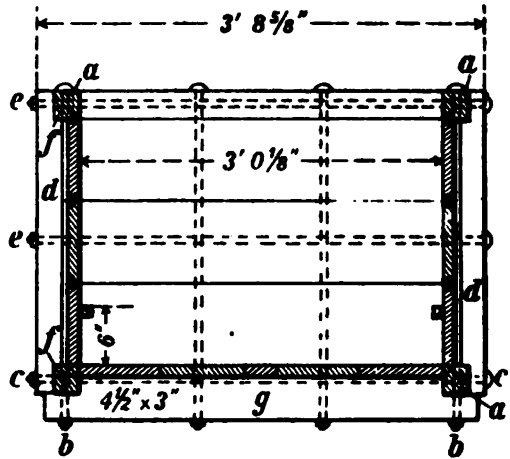
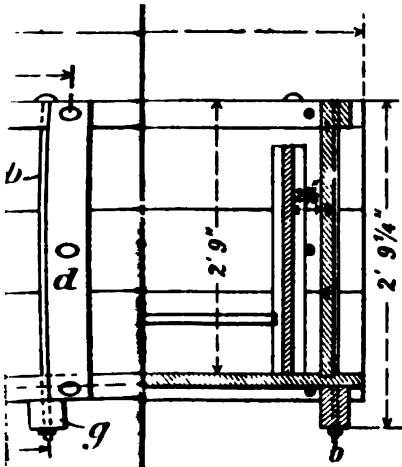
The Pacific Tank Coy. of San Francisco manufacture sheet steel boxes in separate sections of standard size, so that the number of sections is varied according to the quantity of solution to be dealt with. Each section weighs 30 lbs., and contains one narrow compartment for the downward flow of solution, and one large compartment for upward flow, which contains one cubic foot of zinc shavings. An overflow lip is provided for the large compartment, and the sections are placed on a descending series of shallow steps, so that each lip projects over the narrow compartment of the next succeeding section. Two shapes are made, one being rectangular and the other circular in plan, and they appear to be especially suitable for small plants, because a multiplicity of separate vessels is not desirable; and if the units are larger, they may with some advantage be made in the form of vats as described below.

**Zinc Precipitation Vats.**—Reference has been made to the use of vats in cases where precipitation is effected by means of zinc fume, and for this purpose the construction is exactly the same as for an ordinary solution tank, with the addition of pipes for admitting air to agitate the solution and zinc dust.

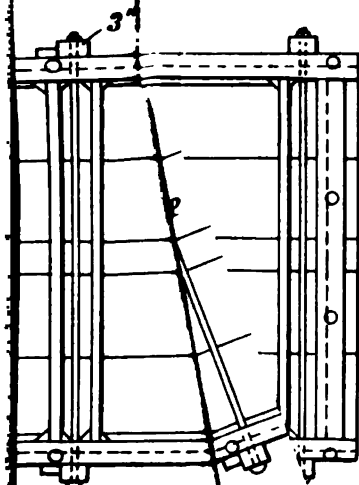
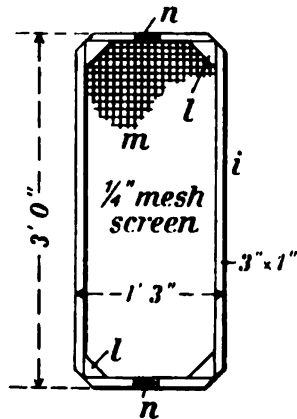
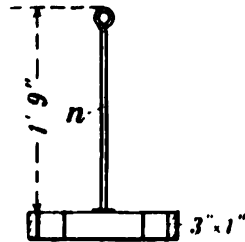
But for ordinary precipitation with filiform zinc, circular vats have been introduced in place of the usual rectangular box by W. A. Caldecott and



[To face page 282.]



*Cross Section.*



*Plan*



P. S. Tavener. In fig. 111 we give an illustration of the type used by Tavener, showing the means adopted to facilitate the clean-up operations.

The vat is made of wood or steel, with vertical sides *a*, conical bottom *b*, overflow trough *c*, and a central vertical pipe *d* for the inflowing solution, which corresponds to the alternate narrow compartments of the rectangular boxes. The solution is carried by a short launder from the circular trough *c* of each vat to the top of the pipe *d* in the next one, so that the direction of flow is upwards through the zinc in every vat. A cross bar at the top and bottom of the pipe carries a bearing for the vertical spindle *e*, which has two blades *f f* under the perforated tray *g*, which supports the zinc shavings. These blades are rotated slowly when cleaning-up the vat, to stir up the gold slime, and to wash it out through the pipe *h*.

#### Electrical Precipitation Boxes.

—The construction of a recent pattern of box for electrical precipitation is shown in figs. 112 to 119. The box is 38 feet long, 6 ft. 1 in. wide, and 3 ft. 1 in.

deep, all inside measurements. It is built of 12 in. by 3 in. Oregon pine, with all joints in outer shell grooved and tongued with clear pine tongues  $1\frac{1}{2}$  in. by  $\frac{1}{2}$  in. Each side is finished off on top with a strip *a*,  $4\frac{1}{2}$  in. by 2 in., laid on the flat. Material of the above dimensions is suitable for practically any size of box that is likely to be required.

There are fifteen pairs of partitions of 1-in. clear pine, grooved and tongued  $\frac{3}{4}$  in. by  $\frac{1}{4}$  in. These partitions are let into grooves  $\frac{1}{2}$  inch deep in the sides of the box to make a water-tight joint, and at the bottom the joints are made good by a triangular fillet. That side of each partition which faces towards the electrode compartments is covered with vertical strips *b b*, 2 ft. 2 in. long by  $3\frac{1}{2}$  in. wide by  $\frac{3}{4}$  in. thick, with  $\frac{1}{2}$ -in. spaces between them and  $\frac{3}{4}$ -in. spaces next to the sides of the box. These spaces form vertical grooves to receive the iron anodes *f f*, which rest on cross pieces of timber *g*, fixed at a suitable height. All the timber is machine-dressed on all faces and edges. The sides are stiffened by upright pieces *c c*, 3 in. square, and the bottom by cross pieces *d d*,  $4\frac{1}{2}$  in. by 3 in. The bolts throughout are of  $\frac{5}{8}$ -in. round iron, screwed at each end, the washers being 2 in. square and  $\frac{1}{4}$  in. thick. There are seventeen horizontal cross pieces *e e* of hardwood, 3 in. wide by 2 in. deep, one being fixed across the top of each narrow compartment and one across

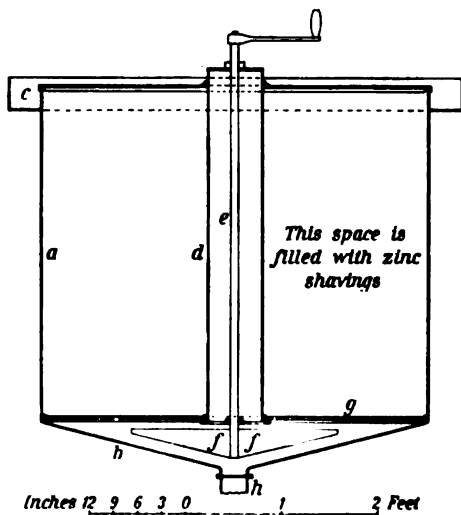


FIG. 111.—Tavener's Circular Zinc Box.

the foot of the box. Each strip has a groove, 6 ft. long,  $\frac{3}{4}$  in. deep, and  $\frac{3}{4}$  in. wide, cut in the top of it, which serves to hold mercury for making the electrical connection of the electrodes.

The iron anode plates are sewn up in hessian, but a tinned iron wire, No. 5 S.W.G., is previously brazed on to one corner of each plate. This wire is so shaped that when the plate is let down the grooves between *b b* to its proper position, the free end of the wire dips into the mercury trough above mentioned. The support for each lead foil sheet (see fig. 116) consists of a single horizontal wire *i*, of the same size, which is bent at the ends so that one end rests in a mercury trough on one side of the large compartment, while the other end of the wire drops in a hole in one of the vertical strips *b b* on the other side of the same compartment.

For the purpose of readily dealing with the deposit of sludge, each compartment is provided with a mudhole *h*, fitted in the following manner (see fig. 119). A 3-in. hole is bored through the side, so low down that it cuts slightly into the bottom, which is also cut to fit a 3-in. flange, care being taken to place the hole so that the cutting for the flange does not go right through the projecting part *j* of the plank. The flange is bolted to the box on the outside of the hole, and fitted with a nipple, socket, and plug. By this arrangement the outlets project sufficiently from the side of the box, so that a trough can be temporarily placed under them when the sludge is to be withdrawn.

A convenient method of supporting such boxes is to let them rest on three longitudinal bearers,  $4\frac{1}{2}$  in. by 3 in., laid flat on the cement floor of the shed, with glazed tiles between these bearers and the cross stiffeners *d d* to ensure good insulation.

**Quantities of Material in Precipitation Boxes.**—For steel boxes the weights generally are, and should always be, given in the makers' catalogues. For wooden boxes, such as those described for zinc and electrical precipitation, a comparison of the actual material in several boxes, of widely different sizes, we find that the quantities may be pretty closely estimated by multiplying the gross contents of the box in cubic feet = *K*, by the coefficients given below.

Let *C* = quantity of timber in box in cubic feet

„ *S* = square feet of timber reduced to 1 in. thick

Then, on the average,

$$C = K \times 0.35$$

The variation of the coefficient, for the cases examined, was only from 0.32 to 0.37, and this variation was independent of the sizes.

For square feet 1 in. thick,

$$S = K \times 4.20$$

For the weight of iron in bolts, nuts, and washers, the coefficient varies rather more.

For the zinc box illustrated, the weight in lbs. is  $W = K \times 1.13$ ; and for the larger electrical box,  $W = K \times 0.9$ .



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Box

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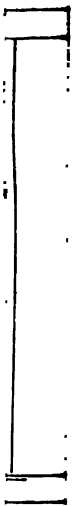
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These figures only apply to solidly built boxes, and not to the flimsy and leaky apparatus too often used in small plants.

**Engines and Boilers.**—It will not be necessary to say very much on this subject, because the former practice of providing a separate power plant for cyanide purposes is very seldom followed at the present time, as it is much more economical to have one central power station for the whole surface work of the mine, and to distribute the power to the various departments by electrical transmission. The following figures may be useful, however, in cases where separate engines and boilers have to be provided for a cyanide plant. The relation between the size of engine and boiler required may be determined by the following simple approximate rules.

Let  $d$  = diameter of engine cylinder in inches

Then for single-cylinder high-pressure engines, actual h.p. =  $d^2 \times \cdot 25$ .

Let  $D$  = diameter of boiler in feet

$L$  = length                    "                    "

Then actual h.p. at 80 lbs. pressure,

For Cornish or Lancashire boilers                    =  $LD \times 0 \cdot 5$

For multitubular or return tube boilers =  $LD^2 \times 0 \cdot 25$

Reckoned in this way, a boiler of any stated power will supply steam for an engine of equal h.p., or several engines aggregating the same amount, provided that they are all near the boiler, that the pipes are well covered, and that the stoking is good. But to provide for irregularities in working, it is as well to allow a margin of 20 per cent. extra boiler power in any case, and much more than this if the steam has to be carried in pipes to engines or pumps at some distance from the boiler, as is sometimes necessary in cyanide works.

**Estimating Power required.**—For sand plants, power may be required for all or any of the following purposes :—

1. Handling the material.
2. Pumping solutions.
3. Driving lathe for zinc cutting.
4. Driving ball mill for grinding lime.
5. Generating current for electrical precipitation.

For slime plants there is also the additional item :—

6. Driving paddles or centrifugal pumps for agitation.

The first item varies so enormously in different cases that no general rule can be suggested, for in small plants manual or animal labour is usually employed, while in large ones it may include the driving of tailings wheels, mechanical haulages, etc. We can therefore only refer to such information as may be found in the chapters dealing with these matters.

For the next three items collectively, viz., 2, 3, and 4, the usual requirements are fully met by the following rule :—For small plants from 2000 to 5000 tons per month, allow 3·5 actual h.p. for each 1000 tons per month, and for large plants 2·5 h.p. per 1000 tons per month.

For electrical precipitation the usual formula is

$$\frac{\text{Volts} \times \text{ampères}}{746} = \text{electrical h.p.}$$

But to allow for loss in conversion, and for belting, etc., it is better to use the following :—

$$\frac{V \times A}{500} = \text{h.p. required from engine.}$$

In slime treatment, the power required for pumping and agitation in small plants of, say, 2000 tons per month, is 14 h.p. for each 1000 tons, and for large plants of 6000 tons and over, 12 h.p. per 1000 tons is a sufficient allowance. It may be roughly estimated that about half of this is actually used for agitation, and the other half for pumping solutions and transferring the pulp.

**Pumps.**—As we have already seen, the power required for pumping solutions is very small, and consequently it is not necessary to consider the relative efficiency of the various kinds of pump from an economical point of view. Of course, with any kind of pump it is better to have an efficient one than an inefficient one, but the selection of the type to be used for any particular part of the work should be made solely on the grounds of convenience and freedom from breakdowns.

The principal factor upon which the type of pump depends, is the position of the main storage vats. (a) If these are above the level of the treatment vats, the solution can be run by gravity to them and through the precipitation boxes into small sumps below the latter. In this case each solution must be continuously and separately pumped back to the storage vats, and therefore one small pump is required for each separate solution (see fig. 125). It is also well to have one spare pump in addition to the three or four in use, so that any one can be replaced if necessary and removed for repairs. For this method of returning the solutions the duplex steam pump of the Worthington type is very suitable, also small centrifugals have been successfully used in some instances. The advantage of the duplex pump is that it can be adjusted exactly to the quantity of solution to be lifted, and will then run for hours without any attention. (b) When the main storage vats are below the precipitation boxes, the only pumping necessary is for supplying solution to the leaching vats from time to time, and for this purpose a pump of large capacity is most suitable (see fig. 122). In this case a centrifugal has many advantages, but as it must sometimes be placed above the storage tanks, it is then liable at times to give trouble in starting, and therefore in this case also a large duplex pump has much in its favour.

In the third case, which is the more usual one in modern plants on flat sites, the storage vats are below, or level with, the treatment vats and above the precipitation boxes, so that both the above systems are required, viz., the continuous lifting from the small sumps to the storage vats, and the intermittent lifting from the latter to the treatment tanks. In this case, for the

last-named operation, the main pump can be placed below the storage vats, and a centrifugal can be used without any fear of losing its vacuum. It is therefore to be preferred on account of its capacity, simplicity, and cleanliness. Only one such pump is absolutely necessary if large enough, because when it is suitably connected, as described in the section on piping, it may be used for pumping each of the different solutions in succession. For instance, the whole of the solutions of a 2000-ton plant can be delivered to six leaching vats with a single 2-inch centrifugal, and the solutions of a 16,000-ton single-treatment plant have been delivered to five leaching vats with equal facility by one 4-inch pump.

For plants with more than, say, 8 vats, and especially for double treatment where there are two sets of vats requiring different cycles of operations, we recommend the provision of two main pumps, each connected to all the solution vats and capable of delivering to any treatment tank. In a double-treatment plant under normal conditions, each pump should supply only one set of tanks, so that the liability to mistakes is minimised; if one breaks down, the other can then do the work of both for the time being.

In slime treatment, centrifugals are almost exclusively employed, both for pulp and solutions, with most satisfactory results, because they suffer less from grit in the slime than any form of pump with valves.

**Capacity of Centrifugal Pumps.**—These are generally catalogued according to the diameter of their outlets; and although their capacity can be made greater by increasing their speed, so that no definite rule can be stated, yet it is better to be satisfied with moderate speed and delivery; not so much because this means greater efficiency, but because it reduces the wear and tear, and consequent liability to frequent stoppages.

The Table LXI. is based on the average practice of several well known makers.

TABLE LXI.—*Capacity of Centrifugal Pumps.*

Diameter of Discharge in inches.	Discharge in Gallons per minute.	Cubic feet per hour.	Short Tons (2000 lbs.) per hour.	Long Tons (2240 lbs.) per hour.
1½	25	241	7·5	6·7
2	50	481	15	13
3	100	963	30	27
4	180	1733	54	48
5	330	3177	99	88
6	500	4813	150	134
7	670	6449	201	180
8	900	8663	270	241

We cannot give any general rule as to the speed at which the several sizes should be driven to obtain the above results, because this depends upon the diameter, which varies according to the maker. It must also be understood

that a delivery of, at least, 50 per cent. more than the above figures can be obtained if desired, and that this table is only intended to represent good practice in relation to cyanide plants.

**Steam Pumps.**—For the low lifts generally required in cyanide plants, the steam cylinder should not be larger than the water cylinder, except when the latter is less than, say, 4 inches diameter, when the steam piston may have from  $1\frac{1}{2}$  times to twice the *area* of the water plunger. With duplex pumps this is not of very great importance, as it means only a certain comparatively small waste of steam when the steam cylinder is larger than necessary, but with single pumps, wherein the main valve is operated by an auxiliary steam cylinder, it is very important to have the right proportions, because otherwise the pump knocks itself about at full load, and is very uncertain in its action when working with less solution at slow speeds.

**Other Pumps.**—In some plants, Manchester donkey pumps have been used with good results. The points in their favour are that they give little trouble, they will run continuously at very slow speeds, are readily adjusted to the variable work required, and take up little floor space, because they can be fixed up against a wall. They are therefore suitable for elevating from sumps below precipitation boxes to the main storage vats. Geared pumps driven by a belt and crank are used in some works for this purpose, but their speed is not so conveniently varied as steam pumps, and if kept constantly working at full speed, are often simply pumping air. They are, however, generally more durable and less likely to get out of repair than steam pumps. One great objection to these pumps is that they require a line shaft to drive them, which is often inconvenient to arrange for.

Whatever type of pump may be selected, all the parts that come into contact with the cyanide solution should be made of iron or steel, and the packing in the stuffing box should be of asbestos or fibre saturated in vaseline or other mineral lubricant.

## CHAPTER XXXVII.

### PIPING, COCKS, LAUNDERS, AND BUILDINGS.

It will be convenient to consider this part of the subject under three heads, namely :—

- (a) The delivery of solution from storage vats to leaching vats.
- (b) The conveyance of the solution from leaching vats to the precipitation boxes, and
- (c) The return of solutions to the storage vats.

In all three cases the greatest attainable simplicity is desirable, in order that the arrangement of pipes can be readily understood and *remembered* by the man on shift. Also, for the same reason, the number of cocks should be as few as possible, and every cock should be placed in a visible and easily *accessible* position. These apparent truisms are often neglected in the design of works.

(a) For putting the various solutions on to the leaching vats, it is well to have large pipes, so that the vats may be quickly filled, and suitable sizes for this purpose will be found in Table LXII. The simplest and therefore best arrangements are shown in figs. 120, 121, and 122. In each figure the storage vats are marked *SS*, and the leaching vats *LL*, while cocks are indicated by small black circles.

In fig. 120, where the storage vats are assumed to be at a higher level than the leaching vats, as in figs. 30 to 33, each of the former has an outlet pipe which leads to all of the latter, and is provided with a branch outlet and cock over each of the leaching vats. Cocks are also placed at *a a*, but these are only necessary when the pipes have to be disconnected for any purpose. They are otherwise superfluous, so that in ordinary working they would remain always open, and may be neglected, in comparing this arrangement with other systems of piping, in respect of simplicity of operation. Under these circumstances, it will be seen that the minimum number of cocks required is equal to the number of leaching vats multiplied by the number of storage vats.

Or if  $N$  = number of leaching vats, and  $n$  the number of storage vats, the number of cocks will be  $Nn = 12$  in fig. 120. It is a good plan to paint

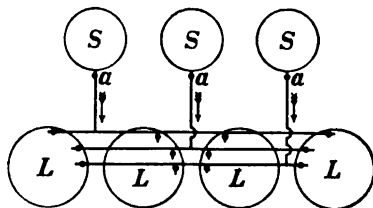


FIG. 120.—Arrangement of Pipes and Cocks.

these pipes different colours, such as red for strong solution, yellow for weak, and blue for alkaline washes. It is not necessary to paint them throughout their length, but only at the delivery cocks, so that the shiftman can make no mistake. This system of piping was at one time almost universally adopted, as it has the advantage that all the solutions can be supplied at the same time to any of the treatment tanks.

In the next system, shown in fig. 121, one large pipe only is used instead of several smaller ones. Thus the solutions can be run on so quickly that the vats can be served in turn by the one pipe for all the different washes. This arrangement requires a little more care and attention on the part of the workman, because at each operation he has to open one cock at *b* as well as one at *c*, and also has to see that all the other cocks are closed, especially the other *b* cocks.

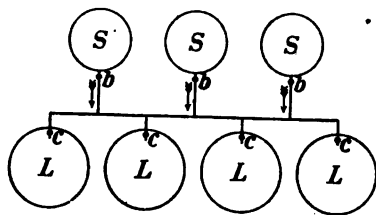


FIG. 121.—Arrangement of Pipes and Cocks.

It has, however, the merit of great simplicity, and using the same notation as before, the total number of cocks required is only  $N+n=7$  in fig. 121. This advantage is more apparent in plants with many vats; thus, with four storage and twelve leaching vats, the first system described above needs forty-eight cocks, and the second only sixteen.

The arrangement shown in fig. 122 is usually adopted when the solutions have to be pumped up to the leaching vats. It is essentially the same as

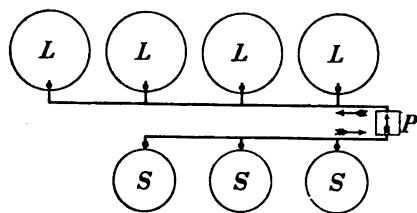


FIG. 122.—Arrangement of Pump, Pipes and Cocks.

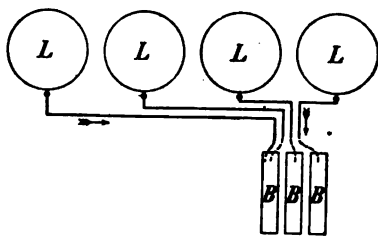


FIG. 123.—Arrangement of Leaching Pipes.

that last described, although it appears slightly different owing to the introduction of the pump *P*. It is important to note that in this case one of the delivery cocks *cc* must be opened *before* the pump is started, and it is a good rule to always leave open the last cock used until the next one required has been opened. Otherwise, with steam pumps, the pipes may burst, or with centrifugals, the belt will be thrown off.

(*b*) For leaching purposes, each vat should have a separate pipe, with a separate *visible* outlet, because the flow of solution from the vat affords the

only evidence as to whether percolation is proceeding in a satisfactory manner. The best plan is to provide a pipe from each vat to the precipitation shed, as in fig. 123. These pipes may terminate close to the precipitation boxes B B, and each may be provided at the end with a piece of flexible hose, long enough to deliver into either of the boxes, as indicated by the dotted lines in fig. 123. Of course, instead of the hose pipes, each leaching pipe can have fixed branches to each precipitation box, but nothing is gained thereby except, perhaps, neatness of appearance, and the method is far less convenient in working.

When leaching into a vacuum chamber, a closed system is, of course, necessary, and in this case a single pipe, with a branch from each vat, with a cock in the branch, is the simplest arrangement.

(c) **Returning Solution to Storage Vats.**—When these are below the precipitation boxes, fig. 124 shows the ordinary system of piping adopted when each box is used for one kind of solution only. No cocks are then required; but if more than one kind of solution is passed at different times through the same box, its outlet pipe must be provided with branches and cocks for the corresponding storage vats.

When the latter are placed above the boxes, the arrangement shown in fig. 125 is suitable. In this figure, boxes and vats are lettered as before, while

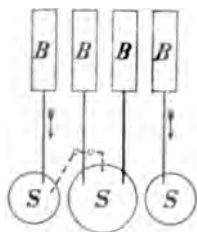


FIG. 124.—Return of Solution to Storage Vats.

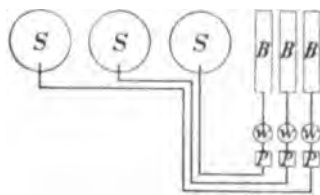


FIG. 125.—Return of Solution to Storage Vats.

*w w* represent small sumps, and *P P* are pumps. In this case also no cocks are required unless, as before, different solutions are passed through the same box, when the same modification is necessary.

Table LXII. gives the sizes of pipes which have been found suitable for the various purposes under the conditions above described.

**Piping for Slime Plants.**—For this purpose such large sizes are necessary that wooden launders are substituted as far as possible, a practice which also assists the aeration of the pulp and solutions.

The principal requirements are—

- (a) Decantation pipes.
- (b) Pulp transfer.
- (c) Residue discharge.
- (d) Solution pipes.

If the decanted liquor can gravitate from the vat, it is only necessary to provide a short outlet pipe discharging into a launder as at *a*, fig. 126. \*This



is connected inside the vat with a decanting pipe *b*, whose upper inlet end *c* is kept by a float just below the surface of the water when working. It is

TABLE LXII.—*Sizes for Solution Pipes.*

System of Piping.	Diameter of Vats.			
	Under 20 feet.	20 to 26 feet.	26 to 32 feet.	32 to 40 feet.
Charging vats by method shown in fig. 120,	2 inch	2½ inch	2½ inch	3 inch
Charging by methods of figs. 121 and 122,	2½ "	3 "	3½ "	4 "
Leaching from treatment vats as in fig. 123,	1 "	1½ "	1½ "	2 "
Returning to storage vats as in figs. 124 and 125,*	1½ "	1½ "	2 "	2½ "

pivoted to *a* by a joint at *d*, similar in principle to that of an ordinary gas bracket, so that it can turn freely as the water level is gradually lowered. When the outflowing liquor has to be raised, the launder *e* may terminate in a sump.

(*b*) For pulp transfer, a pipe system *a a*, shown in thin lines in fig. 127, is required, connected to the bottom of each vat, and through a pump *p* to a

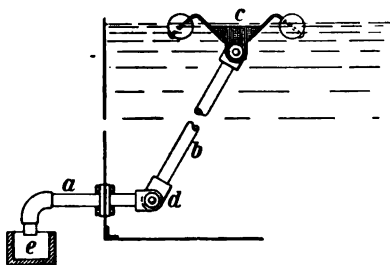


FIG. 126.—Decanting Apparatus.

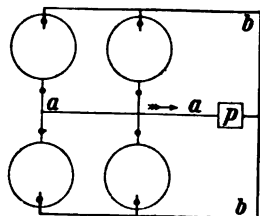


FIG. 127.—Pulp Pipe System.

second system of pipes *b b*, shown in thick lines, which can deliver to any vat.

(*c*) Where discharged residues have to be lifted, the arrangement in fig. 127 will serve for this function also, by providing an additional outlet and cock to the pipes *b b* to deliver the residues into a discharge launder fixed at the required height. When the residues can gravitate to their final resting place, short outlet pipes only are required, which discharge into a common launder, each pipe having a cock or plug. Or the pipes *a a* in fig. 127 can be

\* This last set of sizes depends really upon the size of precipitation box. The figures given are for plants of six treatment vats, and must be increased when more vats are used, unless the number of boxes is increased in proportion.

used for discharging residues, for which purpose they, instead of the pipes *b b*, must be provided with an extra outlet and cock.

(*d*) It is generally possible to use launders exclusively for the conveyance of fresh supplies of solution from the storage vats to the treatment vats, but pipes are usual for returning the precipitated solution from the boxes to the storage tanks.

The following table gives approximately the sizes suitable for the various purposes above mentioned.

TABLE LXIII.—*Sizes of Pipes for Slime Plants.*

Purpose.	Diameter of Vats.				
	Under 20 feet.	20 to 30 feet.	30 to 40 feet.	40 to 50 feet.	50 to 60 feet.
Decantation, . . . . .	2½ inch	3 inch	3½ inch	4 inch	5 inch
Pulp transfer and residue discharge, .	3 „	4 „	5 „	6 „	7 „
Solution return to storage vats, . .	2¼ „	2½ „	3 „	3½ „	4 „

**Fixing Pipes.**—It is convenient when laying pipes to use crosses for connections at right-angled turns, as at *c* in fig. 128 instead of the usual bend shown at *b* in the same figure. The benefit arising from this is, that by removing either of the plugs *d*, the corresponding pipe can be readily cleaned out and obstructions removed. This practical advantage far more than counterbalances any frictional losses which may result from the substitution of the cross for the bend. Crosses can often be used in place of tees for the same purpose. It may be remarked incidentally, that all pipes and fittings in any one plant should be of one make only, as unfortunately the British makers have not yet adopted standard sizes and threads.

Red-lead joints are unsuitable for permanent plants, because the action of cyanide causes leakage after a time. Paraffin wax melted in a little cylinder oil makes satisfactory joints for cyanide solution pipes. The portion of the piping which must be fixed with the greatest precautions is that between leaching vats and precipitation boxes, because it is in this part of the circulating system that the solution carries its maximum value in gold.

All pipes laid below ground level should be placed in wooden troughs of 1½ inch stuff, with removable covers held by screws, the top of the covers being preferably level with the ground.

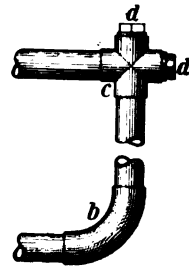
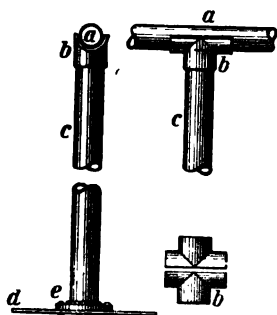


FIG. 128.—Fixing Pipe.

When pipes have to be carried overhead in the open, the device shown in fig. 129 forms a neat and effective support. In this figure, *a* is a 2½-inch solution pipe resting on one-half *b* of a 3-inch cross which has been cut into two pieces, as shown in fig. 130, and which therefore serves for two standards. The half cross is screwed on to the pipe *c*, which is attached at its lower end to a base plate *d* by the flange *e*.

Neat brackets for supporting pipes from sheds or tanks may be made from bends, flanges, and split crosses in a similar manner.



FIGS. 129, 130.—Pipe Carriers.

**Cocks.**—The ordinary plug cocks are often used in cyanide plants, but are not at all suitable, as they soon wear from grit getting between the surfaces, and leak. Where employed, the plugs should be removed regularly, say once a week, and smeared with vaseline or other thick mineral lubricant, and when they show any signs of wear, should be refitted. This is often inconvenient, on account of the position of the cock, and for this reason it is more desirable to use packed cocks with stuffing boxes, such as the Peet valve, which has been largely used in South African plants, or that of the Ludlow Valve Man. Co., which is used considerably in America.

The packing may be of asbestos, but a cheaper and effective material is hemp steeped in molten paraffin wax, thinned with a little cylinder oil. For rich gold solutions the more expensive Kelvin cocks and valves, made by the Palatine Engineering Co. of Liverpool, are to be recommended, but should be ordered with all parts made of iron or steel.

**Launders.**—As the relation between the width and depth of launders is generally a matter of guesswork, it may be worth while to show very briefly how to arrive at a proper practical ratio between these two dimensions.

If *W* be the width of launder and *D* its depth, then the cost is roughly proportional to  $W + 2D$ .

The quantity of pulp passing through when running full is proportional to

$$WD \times \sqrt{\frac{WD}{W + 2D}}$$

And the greatest economy is secured when

$$\frac{\text{Cost}}{\text{Capacity}} = \frac{W + 2D}{WD} \sqrt{\frac{W + 2D}{WD}} = \left( \frac{W + 2D}{WD} \right)^{\frac{3}{2}} \text{ is a minimum}$$

That is to say, when  $W = 2D$ . But in practice the depth of liquid should not exceed two-thirds of the depth of launder, in order to allow for the pulsations which always occur in the flow. Consequently we may say that the best practical shape is obtained when the inside depth is *three-quarters* of the width.

The construction of a good solid launder, such as should be used for carrying solutions, is shown in fig. 131. When several have to be carried side by side, they can, with economy, be combined, as shown in the double launder

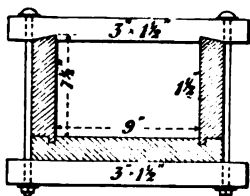


FIG. 131.—Launder.

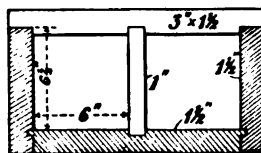


FIG. 132.—Launder.

in fig. 132. The top distance pieces shown in the figures may be from 6 to 8 feet apart.

The minimum grade for launders carrying average tailings is  $2\frac{1}{2}$  per cent., but  $3\frac{1}{2}$  per cent. should be allowed where possible. For slimes from 2 to  $2\frac{1}{2}$  per cent. and for water 1 per cent. is a sufficient grade.

**Buildings.**—For dry crushing plants, and in countries liable to heavy snowstorms for all kinds of plants, it is necessary to cover in the whole of the treatment works. But in places where the climate is favourable, it is now almost universal practice to provide cover only for the precipitation plant, the apparatus for cleaning-up, refining and melting, and all the machinery, while the tanks, classifiers, etc. are built in the open. As in many cases the refinery

TABLE LXIV.—*Floor Space required in Sheds at Cyanide Works, in square feet.*

Tons of Solution precipitated per day.	Boiler Space for Pumping only.	Boiler Space for Electrical Precipitation.	Precipitation and Clean-up with Zinc.	Precipitation and Clean-up with Electricity.	Offices and Stores.	Total with Zinc Precipitation.	Total with Electrical Precipitation.
50 to 100	a	b	c	d	e	a, c, and e	a, b, d, and e
100	270	260	1150	1320	200	1620	2050
200	340	370	1280	1640	240	1880	2610
300	410	480	1420	1970	355	2185	3215
400	480	590	1580	2310	470	2530	3850
500	550	690	1770	2650	590	2910	4480
600	620	800	2000	3000	710	3330	5130

and melting room are quite separate from the cyanide plant, and form an annexe of the general mine laboratory, we shall here consider only the following requirements.

- (1) Boiler-house and machinery.
- (2) Precipitation and clean-up room.
- (3) Storeroom and office.

For the boiler house, a floor of brick or well rammed earth is suitable. For the precipitation room, a concrete floor with a cement facing, finished in neat cement, should be provided, and no expense should be spared in a permanent plant to make this floor as perfect as possible, so that it can be easily swept and kept perfectly clean; otherwise gold will almost certainly be lost when cleaning-up zinc precipitation boxes. The floor should be laid with a slight slope, and should have a small half-round gutter formed in it, suitably placed to intercept all leakage and waste of solution, and to carry the same to a sump, from which it can be pumped through a filter press, and so returned to the storage tanks.

A clear height of at least 12 feet below the tie beams should be allowed in all cases, in order to have plenty of air to dilute the hydrocyanic acid and other gases that escape, and in large sheds this height may be considerably increased with advantage.

The Table LXIV. contains average figures for the floor space required for the various departments of the work for plants of different sizes.

The figures in the above table do not represent the minimum space in which the apparatus can be placed, but allow room for convenient access to every part and for comfortable working. Where power is transmitted electrically to the shed the figures in columns *a* and *b* may be halved.

Plenty of windows should be provided, and good lighting arrangements for night work. The office should contain a bench and shelves for testing work, a desk or table for the log-book, and a chair. It always pays to make the shiftman comfortable and responsible.

TABLE LXV. — *Scantlings for Sheds in Countries free from Snowfall or Abnormal Winds. Dimensions of Timbers and Rods are in inches. The Roof Trusses are of the type illustrated in fig. 188, and are constructed of Timber, except the King and Queen Rods, which are of Round Iron.*

Span in feet.	Posts under Trusses.	Distance apart of Trusses, in feet.	Intermediate Posts.	Runners.	Rafters.	Tie Beams and Angle Struts for same.	Wall Plates.	Purlins.	First Strut at Centre.	Second Strut.	Centre Rod diameter.	Second Rod diameter.	Third Rod diameter.
25	4½ × 3	8	4½ × 3	3 × 3	4½ × 3	4½ × 3	4½ × 3	3 × 2½	4½ × 3	...	1 1/2	1 1/2	...
30	4½ × 3	8	4½ × 3	3 × 3	4½ × 3	5 × 3	4½ × 3	3 × 2½	4½ × 3	...	1 1/2	1 1/2	...
35	5 × 4	9	4½ × 3	4½ × 3	6 × 3	5 × 3	4½ × 3	3 × 3	5 × 4	...	1 1/2	1 1/2	...
40	6 × 4	9	4½ × 3	4½ × 3	6 × 4	6 × 3	6 × 3	3 × 3	5 × 4	4 × 4	1 1/2	1 1/2	...
45	6 × 6	10	5 × 4	6 × 3	7 × 4	6 × 4	6 × 3	3 × 3	6 × 4	5 × 4	1 1/2	1 1/2	...
50	6 × 6	10	6 × 4	6 × 3	7 × 4	6 × 6	6 × 3	3 × 3	6 × 4	5 × 4	1 1/2	1 1/2	...

The nature of the building will depend upon the materials available and the cost. Thus in Siberia log-houses are built for this class of work, as well as for dwellings. In N. America, where the climatic conditions are

also sometimes bad and severe snowstorms occur, heavy construction is required, and suitable rules for building in such cases can be found in engineering books. But in countries where there are no snowstorms or abnormal wind pressures to provide against, a shed framed of comparatively light timbers and covered with galvanised iron is commonly erected, and is quite sufficient for the purpose. As we know of no published data applicable to these cases in a handy form, we give in Table LXV. dimensions of timbers suitable to these conditions, based on actual practice in South Africa, Victoria, and New South Wales. The table will probably be found useful in many other mining districts.

It may be noted that the sizes given for the rods allow for the reduction caused by screwing the ends. If the ends are upset before screwing, the diameter may be reduced accordingly. For covering the shed No. 24 or 26 gauge G.C. iron is almost invariably used, the former being preferable in permanent work. In the United States No. 23 gauge is about the same thickness as No. 24 English.

**Quantities in Sheds.**—These may be roughly estimated in the following manner.

Let  $K$  = gross contents of shed in cubic feet

$C$  = cubic feet of timber in framing

$W$  = weight of rods, nuts, etc. in roof

$w$  = weight of No. 24 G.C. iron.

Then for large sheds or small sheds

$$C = K \times .007 \quad \text{,,} \quad K \times .009$$

$$W = K \times .011 \quad \text{,,} \quad K \times .011$$

$$w = K \times .15 \quad \text{,,} \quad K \times .18.$$

Doors and windows must be reckoned by number and size separately. When there are internal partitions, these can be included in the above approximate quantities by calculating on two or more parts separately, and adding the results together.

## CHAPTER XXXVIII.

### HANDLING MATERIAL.

FROM the commercial point of view, this is one of the most important divisions of the subject of cyanide treatment, on account of the very large quantities of material that have to be dealt with in most cases. The matter may, for general consideration, be conveniently divided into four separate stages, taken in their natural sequence, namely—

- (a) Bringing material to the works.
- (b) Filling vats.
- (c) Discharging vats.
- (d) Dumping residues.

Such of the methods and apparatus employed in these four operations as require discussion in detail will be dealt with separately.

(a) In dealing with very small heaps, the Australian practice is to use Scotch carts, locally known as drays. These are backed up to the heap, filled

by shovelling, then taken to the works and backed, and tipped on to a platform level with the top of the vats. The S. African method of dealing with larger accumulations is, to gradually open out a cutting into the heap for a tram line, and to shovel directly into trucks. When the sand has been piled up to a considerable height, the trucks can be filled so quickly that a single line

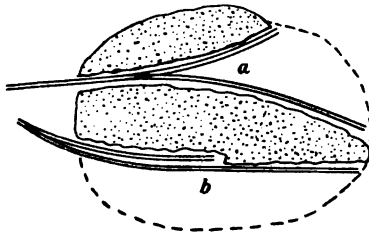


FIG. 133.—Tram Lines.

is generally sufficient. With shallow heaps and in dams, two branch lines are necessary, in order that one set of trucks may be filled while another set is being hauled to the works. In either system, the line is swung round as the material is worked out, so that the trucks are always near the working face. In fig. 133, two ways of arranging the lines are shown. By method *a*, the loading points gradually become further apart, and the workmen have to walk some distance from one to the other, but at *b*, each line serves half the length of the face, and both swing round in the same direction, so that the loading points are always near together, and the work and supervision are more conveniently performed.

Sometimes the material is loosened by ploughing, and is brought to the tram line by means of scrapers drawn by horses in the manner common in

railway construction. With slimy stuff which requires breaking up, this is an excellent method, and is also cheap. In Montana, railroad cuttings are made in the ground and covered by bridges with holes over the track, so that the material brought by the scraper is delivered directly through these holes into trucks.

C. W. Merrill\* says that six men with twelve horses loaded approximately 400 cubic yards per day, with an average haul of not less than 100 feet. In this case, cars of 3-ton capacity with bottom discharge were used on a track with 56-lb. rails, laid to a 3-ft. gauge. A 22-ton locomotive draws sixteen loaded cars up the maximum grade of 3.5 per cent., and the total distance hauled is  $2\frac{1}{2}$  miles. A long line such as this is, however, very unusual in cyanide work, and in most cases side-tipping trucks of about 20 cubic feet capacity are employed. These are preferably drawn by manual labour, animal traction, or ropes worked by a steam winch, when the distances are short and the loading points constantly shifting. When, however, the material can be received and delivered at fixed points, as, for instance, in the transfer from collectors to treatment vats, and the length of the line is considerable, the system of continuous haulage by endless rope is suitable, but only when such *large quantities* are dealt with that the saving in cost of transport is more than sufficient to pay for maintenance of plant and interest on first cost. This remark applies, of course, to all labour-saving machinery, but it is necessary to insist upon it, because of the tendency so often manifested to introduce such devices for the appearance of economy, even in cases where more primitive methods would really be cheaper. And in reference to endless rope haulages, it may be pointed out that even the current costs of transport are not reduced by its introduction when the quantity of material to be shifted is small, because the power absorbed in driving the gearing and rope is out of all proportion to the power consumed in moving the trucks. For short inclined roads which are too steep for animal traction, a steam winch drawing up trains of trucks is suitable, even for small quantities, because the power is directly employed in doing useful work, and the first cost of the installation is small.

For this purpose, winches from 8 to 12 nominal h.p. are generally used with trains of four or six trucks, each of 20 cubic feet capacity, on inclines of, say, 1 in 7. If more trucks are run together, it means not only a larger winch, but also more men for tipping, or a greater loss of time between the trips. The drums should be from 2 ft. 6 ins. to 3 ft. in diameter, and one drum should always be loose on the shaft and provided with a clutch gear. This allows the effective lengths of the two ropes to be altered from time to time as required, so that the full and empty trucks shall arrive simultaneously at their respective stopping-places. It is also most essential that the winch should be amply large enough for the work, and that it should be thoroughly well made and designed, moreover, with a view to the convenience of the driver, a point often overlooked in small winches.

\* The tailings plant of the Montana Min. Co. Ltd.—*Eng. and Min. Journal*.

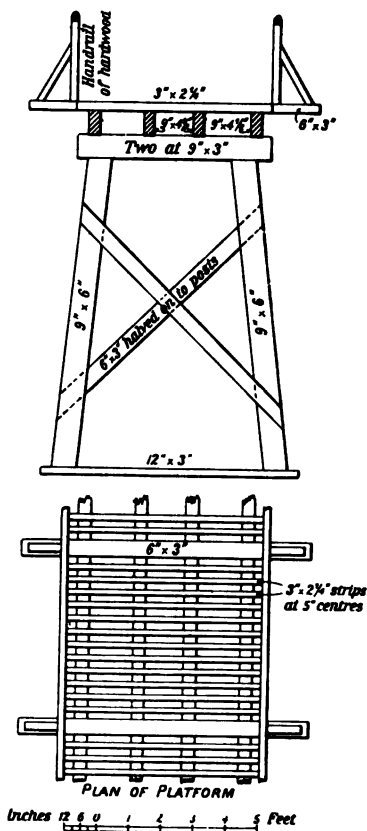


For bringing crushed ore to the treatment vats from extensive heaps, or from dry crushers, and for elevating purposes, belt conveyors are largely used, and other forms of conveyor more rarely (see Chap. XL).

When wet material, such as battery pulp, has to be moved, the obvious method where there is sufficient fall is by launders; and when the vats are

higher than the source of supply, the two practical methods of lifting generally used are by means of tailings wheels and sand pumps, delivering, as the case may be, to revolving distributors, slat gate collectors, or classifiers. In Siberia such material is sometimes lifted by the Archimedean pump, viz., a long cylinder of wood, with a continuous internal thread like a spiral stair. The cylinder is inclined at an angle of about  $40^\circ$ , and the lower end dips into a sump, which is continuously supplied with the pulp. The latter is discharged from the upper end of the cylinder.

(b) The methods of filling pulp into vats have already been sufficiently described. With dry, or nearly dry, material, a good distribution of the material over the whole vat is also desirable, as this tends towards uniformity of texture, and consequently to good leaching. When tipping from trucks, therefore, this should be done from a platform sufficiently high, and so placed as to directly command as much of the vat area as possible, two or more platforms, properly spaced for this purpose, being frequently used. A revolving chute has been employed in connection with belt conveyors in filling vats with dry crushed ore.



FIGS. 134, 135.—Treble Staging for 20-ft. Span.

**Treble Staging over Vats.**—Figs. 134 and 135 illustrate a common form of platform used when tipping from trucks into vats. This gridiron platform has the advantage that it can be made wide enough to provide a firm foothold for the workmen when tipping, but at the same time prevents an accumulation of sand upon the gangway, especially when the spaces between the strips are rather wider underneath than on top. Another form of gangway, in which every alternate strip is cut shorter, is shown in plan figs. 183 and 184 of the complete cyanide plant described in Chapter XLIV. This second form should not be used when Kafirs or other careless labourers are employed, as it is easy for their feet to slip through the open spaces in the

platform. In all parts not directly over the vats, and in all other gangways for workmen, the flooring should consist of 3-inch deals or planks, laid with 1-inch spaces between them.

**Quantities.**—The amount of timber in such staging may be roughly estimated as follows. For a single track averaging 10 ft. above ground level, allow 3 cubic feet per foot run of platform. For double track staging, allow 5.75 cubic feet per foot run. These figures include trestles, bearers, angle bracing, flooring, and hand rails.

(c) The usual methods of emptying vats by hand shovelling and by sluicing with water will be referred to in dealing with working costs, and need no discussion here.

In discharging from excavated tanks at the Langlaagte Estate mine, some years ago, a steam crane placed on rails alongside the vats was employed in the following manner. The body of each empty truck was lifted by the crane and lowered into the vat, where it was filled by shovelling. When full it was again lifted by the crane and replaced upon its carriage, which remained always upon the tram line. Another method still occasionally used is to lay a track upon the floor of the tanks, and to run the trucks into them by large side doors as soon as sufficient sand has been removed. This method is only used for masonry tanks built on the surface of the ground.

The latest novelty for discharging residues is the Blaisdell Rotary Excavator, which is said to discharge at the rate of 100 tons per hour, with an expenditure of only 7 h.p. A description of this will be found at page 185.

(d) For carrying sand residues to dump, side-tipping trucks, drawn by mules for small plants, and by endless rope for large plants, are most common, although a system of conveyor belts is sometimes employed with advantage. When endless-rope haulage is used, the trucks are thereby carried to the highest point of the dump and there released from the rope. After tipping, they gravitate back to the vats, or to a point where they are picked up by the returning part of the rope. As the dump grows in height, the delivery point of the haulage system has to be raised from time to time.

In Western Australia, conveyor belts are used for removing the pressed slime cakes as they are taken out of the filter presses.

#### TRUCKS AND TRAM LINES.

**Trucks.**—In a few cases a 2-ft. gauge has been used in cyanide plants, but experience has shown that the much more common 18-inch gauge is wide enough for all purposes except locomotive work. For small plants a truck holding 16 cubic feet is the usual size, while for large plants 20 ft. capacity is generally adopted. When designing works it is often necessary to know the size and weight of the truck which is to be used. The overall dimensions

for 18-in. gauge side-tipping V-shaped wagons are given in figs. 136, 137, 138, 139, and the corresponding weights are as below.

20 cubic ft. capacity with 15-inch wheels. Each 800 lbs.						
16	"	"	"	15	"	" 720 "
16	"	"	"	12	"	" 610 "

The use of 12-inch instead of 15-inch wheels for the smaller truck reduces the overall height to 39½ inches.

These sizes and weights are for trucks which we have used largely on the

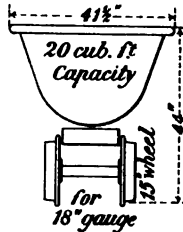


FIG. 136.—Truck.

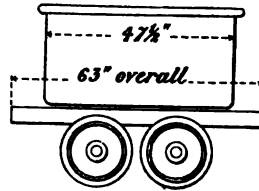


FIG. 137.—Truck.

Witwatersrand and in Australia, and the figures were kindly supplied by the maker, R. Hudson, of Gildersome Foundry, Leeds. Generally speaking, the weight of trucks from 15 to 60 cubic ft. capacity varies from 24 to 36 per cent. of the load carried. The mean of these two figures, or 30 per cent., may be taken to represent the lightest class of trucks allowable for work of this kind, where they often have to undergo a great deal of rough handling.

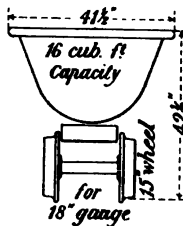


FIG. 138.—Truck.

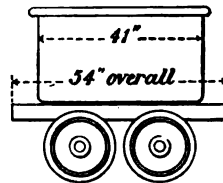


FIG. 139.—Truck.

**Tram lines.**—For temporary tracks that have to be frequently shifted, steel sleepers are undoubtedly the best, but for permanent lines, wooden sleepers 6 ins. by 4½ ins. in section and about 3 feet long give a better hold on the ground. Also, on curves of short radius, wooden sleepers should be used, because they easily allow of the slight increase of gauge which is desirable, while with steel sleepers this alteration involves some trouble and expense. Of course, sharp curves must be avoided as far as possible, as they cause enormous wear of axles, bearings, and wheel flanges. The minimum curve that can be practically used at a pinch with 18-inch gauge is about 12-foot radius, but this is very bad practice unless space is absolutely limited,

and a minimum of 30-feet radius should be aimed at in laying out permanent lines.

As regards size of rail, 12 lbs. per yard is the lightest that should be used, and is about good enough for shifting lines on dumps and the like, while 14 or 16 lbs. are suitable for permanent work. The width of the bottom flange ought to be about the same as the depth of the rail. A lot of light rails are marketed whose depth is much greater than their width, but these should be carefully avoided, because in light rails, lateral stiffness is quite as important as vertical carrying power. This may seem to be a minor detail, but it makes considerable difference in practice. In laying these lines, the track should be well stretched; that is to say, after every two or three lengths have been connected, they should be forced apart as far as the holes in rails and fish-plates will allow.

In laying out tracks, the following points should be kept in mind. For animal traction, uniformity of grade is of very great importance, and in order to combine this with cheap construction, the best method is to lay out on the natural surface of the ground, the curved line which is found by trial with the levelling staff to correspond to the grade selected.

For continuous-rope haulage, the chief point is to make the line as straight as possible, moderate changes of gradient being relatively much less important. If the track cannot be laid in one straight line, then it should be made up in long straight lengths, with short definite curves at the places where the direction changes.

**Tractive Force.**—We have found in working 18-inch tram lines that a train of six or eight well-greased full trucks will just move by their own weight on an incline of 1 in 70, while empty trucks require a slightly steeper grade. This may therefore be considered as a practical measure of the minimum force necessary to overcome the rolling friction.

Let  $W$  = total weight of trucks and load in lbs.

$T$  = tractive force in lbs., and

$K$  = coefficient of rolling friction.

Then, on an incline, the tractive force necessary to lift the load, or, conversely, the force exerted by the load in running down, is correctly expressed by the equation

$$T = W \sin \alpha \quad (1)$$

where  $\alpha$  is the angle of inclination.

But in cyanide plants the steepest grade is seldom more than 1 in 5, so we can substitute the more convenient expression,

$$T = W \tan \alpha \quad (2)$$

with an error of only 2 per cent. in the extreme case in excess of the true quantity. But  $\tan \alpha$  is simply the fraction in which gradients are usually expressed. That is to say, for a grade of 1 in 7, the symbol  $\tan \alpha$  is  $\frac{1}{7}$ . Therefore, calling this fraction  $G$ , in any case we have the simple formula

$$T = WG \quad (3)$$

Consequently, on the grade of 1 in 70 mentioned above, the tractive force which just overcomes the friction is  $\frac{W \times 1}{70}$ . This, then, is the minimum value of the coefficient of friction  $K$ . But as trucks and rails are not always in good condition, it is better to make  $K = \frac{1}{70}$  in calculations for ordinary practical cases.

Finally, then, the total tractive force required in working on an incline is

$$T = W(K \pm G) \quad . \quad . \quad . \quad . \quad . \quad (4)$$

The plus sign is used when pulling up, and the minus sign when pulling down the gradient.

**Equalising Tractive Force.**—When, as in figs. 30 and 31, a residue dump is to be formed at a lower level than the treatment vats, and, generally, when material has to be moved downhill by animal traction, it is advantageous to lay out the grades so that the pull for loaded trucks down the incline is equal to the pull required for the empties up the incline. When a single line is used, and whenever the outgoing and return lines are of the same length, the grade necessary for equal tractive force in either direction can be found as follows—

$W$  = weight of loaded trucks, and  $w$  = weight of empties.

Then, pull down the incline =  $W(K - G)$ , and pull up for empties =  $w(K + G)$ .

Equating these, and transposing, we have

$$G = \frac{K(W - w)}{W + w} \quad . \quad . \quad . \quad . \quad . \quad (5)$$

Taking the weight of a 20-ft. truck at 800 lbs., and its contents as 105 lbs. per cubic feet, including moisture, the required grade is 1 in 88.

In some cases the return line is shorter than the down line, and must therefore have a steeper gradient in order to reach the starting-point of the down line at the proper level. When the down grade is  $n$  times the length of the up grade, and  $G$  is the grade of the down line, then the returning gradient must be  $nG$ , and the equation for equal tractive force on both is—

$$G = \frac{K(W - w)}{W + nw} \quad . \quad . \quad . \quad . \quad . \quad (6)$$

To further facilitate the selection of suitable grades, the following table has been worked out from the above equations for the 16- and 20-ft. trucks, whose weights have been already given.

On short circular tracks, such as those from collecting vats to treatment vats in fig. 35, and from treatment tanks to residue dump in fig. 37, it is better to make the grade steeper than is given above when manual labour is employed, because men work better intermittently. In fact, men prefer to work on a grade at which the loaded trucks will run down by their own weight, although this necessarily involves harder work in pushing the empty trucks back again. A gradient of 1 in 50 or 60 is therefore suitable for the full trucks in such cases.

TABLE LXVI.—*Showing Grades required for equal tractive effort on outgoing and return lines, when loaded trucks run on the down gradient.*

Weight of Truck in lbs.	Cubic feet carried.	Fractions given below show down grade = G when $K = \frac{1}{4}$ .									
		Load at 90 lbs. per cubic foot.					Load at 110 lbs. per cubic foot.				
		n = 1	n = 2	n = 3	n = 4	n = 5	n = 1	n = 2	n = 3	n = 4	n = 5
800	20	1/94	1/117	1/139	1/161	1/183	1/86	1/105	1/123	1/141	1/159
720	16	1/100	1/125	1/150	1/175	1/200	1/91	1/111	1/132	1/152	1/173
610	16	1/92	1/114	1/135	1/156	1/177	1/85	1/102	1/119	1/137	1/154

**Engine Planes. Power.**—These may be worked at a speed of six to ten miles an hour. In cases where an extra steep grade occurs on a very small portion of the whole length, a speed of four miles may be allowed for, on this portion only, when calculating the power which has to be provided. Of course, where very small quantities are handled in this way, slower speeds may be used in order to economise in the cost of working the hauling winch.

The h.p. required in any given case is found by multiplying the tractive force in lbs. for the steepest gradient on the line, by the speed in feet per minute on that gradient, and dividing by 33,000.

Thus if W = total weight in lbs., including trucks and rope

G = as before, grade expressed as a fraction

M = speed in miles per hour, and

P = actual h.p. to be provided, then

$$P = 0.0027MW(K + G) \quad (7)$$

It is to be noted, however, that while the maximum grade is used to determine the necessary power of the winch, only the average grade is to be considered when calculating the mean power expended during the whole time of hauling.

So far, we have only considered a single set of trucks drawn up the incline, but when a set of empties is lowered at the same time, this will lessen the work of the winch. Let  $p$  = power derived from fall of empty trucks whose weight is  $w$ . Then

$$p = 0.0027Mw(G - K) \quad (7a)$$

and the total nett h.p. is therefore

$$P - p = 0.0027M\{W(K + G) + w(G - K)\} \quad (8)$$

In most cases, however, it is better, when deciding upon the power to be provided, to neglect altogether any gain from this source, and to make provision for the maximum possible load by using equation (7).

On flat inclines, where a tail rope has to be used, equation 8 is applicable also, and should be employed for double lines with two sets of trucks, because in such cases the empty trucks add to the power required from the engine.

In any case, at least 10 per cent. extra power must be allowed for engine friction, starting the trucks, and other matters which do not admit of close calculation. When the winch is situated at a distance from its boiler, the consequent loss of pressure in the steam pipes must also be taken into account.

**Power for Endless-Rope Haulages.**—Equation 8 is also useful in calculating the power required for continuous-rope haulage when the weights upon the up and down grades respectively can be accurately ascertained, but it only applies strictly to the trucks and their contents.

However, in this case the friction of the rope, driving gear, and road rollers becomes an important item, which may often account for more than half the total power. Some engineers recommend the estimation of this by taking the total weight of all ropes, pulleys, and other moving parts, and calculating in the same way as for the trucks, but with a coefficient of  $\frac{1}{25}$  instead of  $\frac{1}{40}$ . The only safe course is to have plenty of spare engine power, which, after all, only adds very slightly to the first cost of the whole installation, and it also provides for any temporary abnormal loading, due to variations in the supply of trucks to the line.

## CHAPTER XXXIX.

### ROPES AND GEAR FOR HAULAGE.

OF the many systems of rope haulage which are used for different purposes, only the two already referred to are usually of service in connection with cyanide works, namely, inclined roads worked by steam winches and continuous-rope haulage, with the rope usually *over* the trucks. Consequently we shall only deal with those points in connection with ropes which are of importance in the use of these two systems.

Steel wire ropes are usually made of either—

1. Bessemer steel, with a breaking stress of about 35 tons per square inch of the section of the rope.

2. Siemens-Martin steel, with a breaking stress of about 50 tons per square inch ; and

3. Crucible cast steel, with a breaking stress of 90 tons per square inch.

The last is the best for the purpose in view. It is evidently of importance, in ordering rope, either to specify the material of which it is to be made, or to have a guarantee as to its strength.

We shall see later, in considering the stresses in the rope caused by bending it, that in most cases it is desirable to have a rope made of small wires and of small sectional area. The principal reason why wire of great tensile strength should be used, is that the rope and its constituents may be as small as is possible, consistent with the necessary provision for the wearing down of the outer wires by friction against the drums and sheaves.

Another reason for employing wire of great tensile resistance is that a rope of given strength is lighter than a rope of equal strength made of inferior material.

For haulage purposes, ropes are usually made of six strands round a central core of hemp, and each strand contains either 7, 12, 19, or 24 wires.

Table LXVII. gives the breaking weight of such steel wire ropes, for all the sizes likely to be required for our purpose, calculated at 90 tons per square inch. The breaking weights are given in lbs., so that they may be readily converted either into long or short tons.

**Working Stress on Rope.**—This consists of two principal items, first the tractive pull found in the manner already described in Chapter XXXVIII., and, secondly the stress due to the bending of the rope round the winding drum or the sheaves and pulleys on the track. This second item is a very important



one, but is too often neglected in practice, or rather it is allowed for in a rough and ready way, either by taking a large factor of safety, or by some empirical rule as to the ratio between the diameter of the rope and the minimum diameter of sheave which may be used. But as this bending stress is

TABLE LXVII.—*Showing calculated Breaking Weights in lbs. of Steel Wire Ropes at 90 long tons per square inch.*

Standard Wire Gauge number.	Diameter of Wire in decimals of an inch.	6 Strands, each of 7 Wires.	6 Strands, each of 12 Wires.	6 Strands, each of 19 Wires.	6 Strands, each of 24 Wires.
25	·020	...	...	...	9,529
24	·022	...	...	8,735	11,034
23	·024	...	6,567	10,397	13,133
22	·028	...	8,939	14,153	17,868
21	·032	6,809	11,673	18,482	23,346
20	·036	8,619	14,775	23,394	29,550
19	·040	10,640	18,240	28,880	...
18	·048	15,322	26,267	41,589	...
17	·056	20,855	35,751	...	...
16	·064	27,239	46,695	...	...
15	·072	34,474	...	...	...
14	·080	42,560	...	...	...

capable of fairly exact calculation, it is far better to ascertain it, and thereby to know more accurately the actual total stress which is being put upon the rope in any given case.

**Stress due to Bending.**—When a wire rope is bent over a pulley, as shown in fig. 140, it is clear that although the centre line *ab* of the rope retains its original length, the wires on the outer side *cd* must be lengthened, and those on the inner side *ef* must be shortened. That is to say, there is a tendency to set up a tensile stress in the outer wires and a compressive stress in the inner wires. But the wires of each strand are twisted round the strand, and the strands are twisted round the rope so that each wire occupies alternately the inner and outer position. Consequently the tendency to extension and compression occur alternately in the same wire and neutralise each other, thus, as Weisbach has pointed out,\* increasing the probability that the tension is uniform in all the wires. But, although there is probably no stress due to the bending of the rope considered as a whole, there is a stress set up in each wire due to the bending of the wire itself. Thus, if we now consider fig. 140 to represent the bending of a single wire, it is evident that the outer fibre *cd* will be in tension and the inner fibre in compression. The former will be added to the tension already existing in the wire due to its share of the tractive force, and the sum of these two



FIG. 140.—Stresses on Ropes.

\* *Mechanics of Engineering*, vol. iii, part i., 1890, p. 556.

tensions will represent the maximum stress on the wire. It is true that this maximum stress is only exerted on the outer fibres of each wire, but it is also true that if these outer fibres are overloaded, the whole wire will be destroyed.

The stress  $S$  in a wire of diameter  $d$  produced by bending in the arc of a circle of diameter  $D$  is given by the expression

$$S = \frac{Ed}{D}$$

where  $E$  is the modulus of elasticity of the steel.\* The two diameters  $d$  and  $D$  must of course be expressed in the same units, such as feet or inches, and the stress will be given in the same units as the modulus of elasticity.

This formula brings out two important points very clearly, viz.—

1. That the stress in the wires caused by bending is inversely proportional to the diameter of the sheave round which the rope is bent, and therefore the largest sheaves that are practically convenient should be used.

2. That the stress is directly proportional to the diameter of the wire of which the rope is composed, and consequently, if for any reason it is necessary to use small sheaves, a rope composed of comparatively small wires is to be preferred in such a case.

To apply the above formula in practice for any given rope we must first find the stress ( $S$ ) per square inch, and then multiply the figure thus found by the total sectional area of all the wires of the rope, expressed in square inches.

One example will make this quite clear, and we may take the case of a rope which consists of 6 strands of 7 wires, each wire being of No. 16 S.W.G., that is, .064 inch in diameter. Assuming a 3-ft. pulley and taking  $E = 29,000,000$ , the above formula becomes

$$S = \frac{29,000,000 \times .064}{36} = 51555 \text{ lbs. per square inch}$$

The sectional area of each wire is .003217 square inch, so the total section of the 42 wires = .135114.

$$\text{And } 51555 \times .135114 = 6914 \text{ lbs.}$$

In the Tables LXVIII. to LXX. these stresses will be found for all the sizes of rope included in table LXVII., and for the sizes of pulleys which are likely to be used in practice.

In view of the important relation which exists between the bending stresses and the size of the individual wires, it is unfortunate that makers of wire ropes, as a rule, give no information as to these sizes in their catalogues, but are content to state only the circumferences of the whole ropes, which, of course, vary with each particular make. For instance, one maker catalogues a rope of 42/19 as  $1\frac{3}{4}$  inches circumference, and another maker has a rope of 42/20 described as  $1\frac{3}{4}$  inches.

\* For proof of this formula in the form, tension =  $\frac{Ed}{2R}$ , see *Transmission of Power by Wire Ropes*, by Albert W. Stahl, M.E., New York.

TABLE LXVIII.—*Bending Stresses in Wire Ropes composed of Six Strands, with Seven Wires in each strand.*

Diameter of Sheave.	21 S. W. G.		20	19		18	17		16	15		14	No. S. W. G.	
	.032			.040			.048			.056			.064	
6 ins.	5224		7439	10208		17632	...		...	...		...	These numbers represent total stress in pounds due to bending.	
1 ft. 0 "	2612		3719	5104		8816	14000		20896	...		...		
1 " 6 "	1741		2480	3403		5374	9333		13932	22401		27210		
2 " 0 "	1306		1860	2552		4408	7000		10449	16801		20408		
2 " 6 "	1045		1488	2042		3526	5600		8359	13441		16326		
3 " 0 "	871		1240	1701		2939	4667		6914	11201		13605		
3 " 6 "	...		1063	1458		2519	4000		5971	9601		11661		
4 " 0 "	...		930	1276		2204	3500		5244	8401		10204		
4 " 6 "	...		...	1134		1948	3111		4644	7467		9070		
5 " 0 "	...		...	1021		1763	2800		4180	6720		8163		
5 " 6 "	...		...	...		...	2545		3800	6109		7421		
6 " 0 "	...		...	...		...	...		...	5600		6803		

TABLE LXIX.—Bending Stresses in Wire Ropes composed of Six Strands, with Twelve Wires in each strand.

Diameter of Sheave.	23	22	21	20	19	18	17	16	No. S. W. G.
	.024	.028	.032	.036	.040	.048	.056	.064	Diameter of Wire in inches.
6 ins.	3778	6000	8956	12752	17492	...	...	...	These numbers represent total stress in lbs. due to bending.
1 ft. 0 "	1859	3000	4478	6376	8746	15099	23999	...	
1 " 6 "	1259	2000	2985	4251	5331	10066	16000	23883	
2 " 0 "	945	1500	2239	3188	4373	7550	12000	18912	
2 " 6 "	756	1200	1791	2550	3948	6040	9600	14330	
3 " 0 "	630	1000	1493	2125	2915	5033	8000	11942	
3 " 6 "	...	...	1279	1822	2499	4314	6857	10236	
4 " 0 "	...	...	1119	1594	2187	3775	6000	8956	
4 " 6 "	...	...	...	...	1944	3365	5333	7961	
5 " 0 "	...	...	...	...	1749	3020	4800	7165	
5 " 6 "	...	...	...	...	...	2745	4364	6514	
6 " 0 "	...	...	...	...	...	2517	4000	5971	

Note.—For ropes composed of six strands of twenty-four wires in each strand, the stresses will be just double the figures given in this table.

TABLE LXX.—*Bending Stresses in Wire Ropes composed of Six Strands, with Nineteen Wires in each strand.*

Diameter of Sheave.	24	23	22	21	20	19	18	No. S. W. G.
	•022	•024	•028	•032	•036	•040	•048	Diameter of Wire in inches.
6 ins.	4608	5988	9501	14180	...	...	...	
1 ft. 0 "	2304	2991	4750	7090	10096	13848	23930	
1 " 6 "	1536	1994	3167	4727	6730	9232	15953	
2 " 0 "	1152	1496	2375	3545	5048	6924	11965	
2 " 6 "	922	1197	1900	2836	4038	5539	9572	
3 " 0 "	768	997	1583	2363	3365	4616	7977	
3 " 6 "	...	855	1357	2026	2884	3956	6837	
4 " 0 "	...	748	1178	1773	2524	3462	5982	
4 " 6 "	...	...	...	1576	2243	3071	5322	
5 " 0 "	...	...	...	1418	2019	2770	4785	
5 " 6 "	...	...	...	...	...	2518	4351	
6 " 0 "	...	...	...	...	...	2808	3999	

These numbers represent total stress in lbs.  
due to bending.

**Factor of Safety.**—When the maximum working load on the rope and the stresses due to bending have been carefully ascertained, it is not necessary to use such large factors of safety as are required when the loads and stresses are only roughly estimated. In the majority of cases in connection with metallurgical works a factor of safety of 4 is amply sufficient for the working load. In cases of inclines, where any danger to workmen would result from the breaking of the rope, a factor of safety of 5 or 6 should be allowed. In all cases a factor of 3 is sufficient to allow on the calculated bending stress.

### *Examples.*

A few examples will illustrate the most convenient method of using the above tables.

*Example 1a.* In a small plant, 2 trucks of 16 cubic feet capacity are to be pulled up a maximum incline of 1 in 9.

If each truck weighs 650 lbs. and its contents 1600 lbs., the total load will be 4500 lbs. Allowing for friction at  $\frac{1}{10}$  of the load, and for the lifting of the weight up the incline, the working load on the rope will be 590 lbs. Multiply this by 5 as the factor of safety, and the product 2950 will be the strength required in the rope in respect of the load upon it. For a rope with 7 wires per strand, Table LXVII. gives the breaking weight for No. 21 gauge at 6809 lbs. Then the balance of this ( $6809 - 2950 = 3859$ ) is available to resist the bending stress, or, in other words, we may allow a bending stress equal to one-third of this, say 1286 lbs. Now, turning to Table LXVIII., we find that this number lies between those in the column headed 21 gauge, which correspond to a 2 ft. and a 2 ft. 6 in. sheave. We conclude, therefore, that a rope of 6 strands of 7/21 could be safely used under the given conditions on a sheave 2 ft. 3 ins. in diameter.

1b. Assuming that we wish to use a still smaller sheave, we take the next larger size of rope, namely 7/20, which, according to Table LXVII., has a breaking strength of 8619 lbs. Subtracting as before 2950 „

we have an excess of 5669 „, which, divided by 3, gives an allowable bending stress of 1890 lbs. This is a little greater than the figure for a 2 ft. sheave, which can therefore be safely used.

Thus we find that for a given working load and a given small-sized sheave, a rope of larger diameter gives better results than a small one, at all events within certain limits. But as a larger rope is heavier to use, we will, as a further extension of this same example, see what is gained by the use of a rope which has 12 wires in each strand.

1c. As before, we select from Table LXVII., but in this case from column 4, a rope with a breaking load greater than 2950. As a first trial take 12/23 = 6567 lbs. Subtracting 2950 gives 3617, which divided by 3, gives 1206 lbs. as the permissible stress due to bending. Turning now to Table

LXIX. column 2, we find that a 1 ft. 6 in. sheave causes a stress of 1259 lbs., and that therefore a sheave slightly larger than this, say 1 ft. 7 in., will be sufficient for this given rope and load.

Collecting the results of these three cases, we have as suitable ropes for the purpose—

Rope of 42/21 working on a minimum sheave of 2 ft. 8 in.			
„ 42/20	„	„	2 „ 0 „
„ 72/23	„	„	1 „ 7 „

*Example 2.* Suppose a rope to carry a direct pull of 2500 lbs. over a minimum sheave of 4 ft. diameter. Assuming first a 42 wire rope.

Direct load  $2500 \times 4 = 10,000$  lbs. = breaking strength to resist direct pull.

From Table LXVII. column 3, select a likely looking number, say 20855, corresponding to 42/17. Then, subtracting 10,000 lbs., we have 10,855 lbs. to resist bending stress, or an actual allowable stress of 3618 lbs. Table LXVIII. shows in column 6 that such a rope over a 4-ft. sheave will be subject to a stress of 3500 lbs., and is therefore suited to the conditions assumed. Of course, one may not always select the right size from Table LXVII. at the first attempt, but a glance at the table of bending stresses shows whether it is right or wrong, and in any case only two or three trials are necessary.

It is, in fact, far more convenient to use the tables than to construct and use a complicated formula, such as would be necessary to give the required size at once.

It has been pointed out \* that in cases where the rope is subjected only to a very slight stress, the tension in the rope may be insufficient to bend it to the actual curve of the smallest sheave used, and that in such cases the size of the sheave is of no consequence, as the bending stress depends upon the actual curve of the rope itself. But in the applications of rope haulage which we are now considering, it may be fairly assumed that the loads put upon the rope in practice will be quite large enough to bend the rope to the smallest sheave, found by means of the above tables in the manner described. Rollers on which the rope lies slack may be, and are in practice, made smaller than the minimum sheave.

**Gear.**—The first requirement common to all systems is the provision of a number of horizontal rollers on the track between the rails, to prevent the rope from dragging on the ground or on the sleepers. A convenient form of roller for this purpose is one of cast iron, 4 or 5 inches in diameter and about 11 inches long, with end flanges. It may be supported in hard-wood bearings, bolted down to a sleeper, which may be one of the ordinary track sleepers (see fig. 141). These rollers should not be more than 6 yards apart, and they

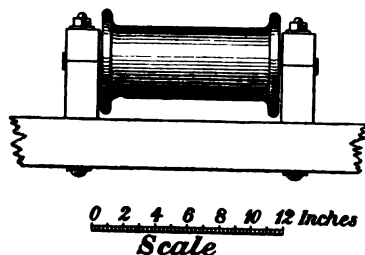


FIG. 141.—Roller.

\* *Engineering Association of South Nashville, Tennessee, Proc.*, vol. vii. p. 81.

must, of course, be fixed and sized so as to be clear of trucks and wheels. At all points where there is a convex change of grade, as at  $S_1$  in fig. 36, a larger roller must be fixed, whose diameter is equal to the minimum calculated by the rules given above.

On roads worked by hauling engines, in passing round curves, a series of similar rollers fixed vertically is used to guide the rope. These should always be of the calculated minimum diameter, because the rope is under tension when pressing against them. This point is often neglected.

In endless-rope systems, the most important points are, means for giving motion to the rope, for maintaining the tension approximately constant, and for attaching the trucks.

A curved drum, such as shown in fig. 142, is a simple and convenient

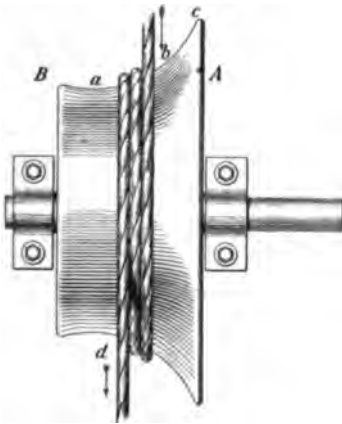


FIG. 142.—Curved Drum.

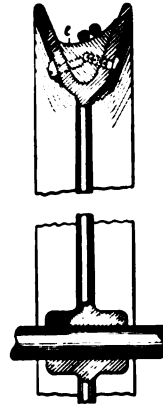


FIG. 143.—Driving Wheel.

0 1 2 3 4 5 6 7 8 9 10 11 12 Inches.

Scale for Figs. 142, 143, 145, 146, 147.

device for driving purposes. The curve  $abc$  is so laid out that at one point, say at  $a$ , the surface of the drum is parallel to its axis, while at  $c$  it is nearly at right angles to it, so that it passes through every intermediate angle between  $a$  and  $c$ . The rope is led to the drum at  $b$ , as shown by the arrow near that end of the drum, which has the largest diameter, and after several complete turns it passes off at  $d$ . It is clear that the continual feeding on at the right-hand side, and paying off at the left-hand, will tend to cause the rope to climb up towards the flange. But this is counteracted by the tendency of the coil as a whole to slip sideways towards  $B$ . Climbing towards  $A$ , however, brings the rope to a steeper part of the drum, and therefore increases the side slip, while motion towards  $B$  brings it to a flatter part, where the side slip is less. Consequently, when either tendency prevails for



a time, it automatically shifts the coil on the drum, until the two opposing forces are again balanced. With the drum shown, the rope can be driven in the opposite direction by taking it off and coiling it on in the opposite sense. Or a drum with the curve *abc* repeated on the left-hand side may be used to drive in either direction at will. Wide drums like the one illustrated have been used, but are not necessary when the speed of working is constant, because the actual change of position of the coil is not great. A wheel of the form illustrated in fig. 143 is therefore quite suitable. The chief objection to this method of driving is the wear and tear caused to rope and drum by the side slip. With proper lubrication this objection is not serious, and may be lessened as regards the drum by forming its wearing surface of separate replaceable blocks of cast iron, hard wood or rubber, as at *e* in fig. 143. For very elaborate installations, the usual arrangement of tandem V-wheels, as employed for street tramways, may be used with advantage.

The ordinary arrangement of tightening gear is shown in fig. 144. It con-

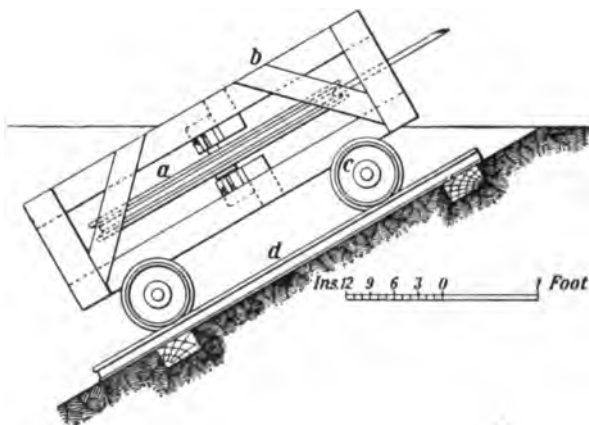


FIG. 144.—Tightening Gear.

sists of a V-wheel *a*, fixed in a frame *b*, supported on wheels *cc*, which run on a steeply inclined track *d*. A second frame loaded with weights may be attached to *b*, or one frame may carry both wheel and weights. This apparatus is preferably applied to the rope directly it leaves the driving gear.

For overhead ropes, at the points where the trucks come on to the rope, and where they leave it, overhead pulleys are provided, placed sufficiently high to allow a truck and its grip to pass beneath them. The rope hangs, as it were, in a long festoon between these pulleys, from one end of the line to the other, the lowest part resting upon the road rollers. The truck is pushed forward until it reaches a part of the rope low enough to catch in the grip, the latter being properly placed to receive it. At the delivery end of the line, as the rope rises, it simply lifts out of the grip and leaves the truck free.

Sometimes the same arrangement of overhead rollers is employed at curves

to release the truck, which then gravitates round the curve. The rope, after passing round a horizontal sheave into its new direction, and over a second roller, is allowed to descend again, and carry the truck forward. This arrangement requires a man to attend to the grips at the point where they pick up the rope again. A similar device is also used for transferring trucks to and from branch lines.

Another method of dealing with curves is to place a horizontal V-pulley exactly at the height of the rope when in the grip, and with its outer rim just clear of the grip. As a truck passes, the rope is momentarily drawn out of the V, but it springs back by its own tension as soon as the grip has passed the pulley. Although this device is nominally self-acting, it requires supervision, lest the rope should leave the V-pulley when there is a long interval between two trucks.

A very common form of grip for attaching trucks to rope is shown in fig. 145, the dimensions there given being suitable for a  $\frac{3}{4}$ -inch rope. The sockets *aa* on the end of the truck should be fixed about 5 inches from the centre

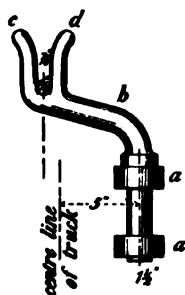
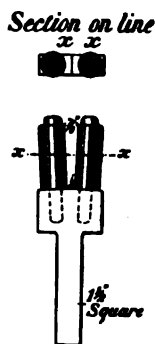


FIG. 145.—Grip.



FIGS. 146, 147.—Grip.

of the truck. When the rope falls into the V between *c* and *d*, the grip turns in the socket and jams firmly on the rope, which is thereby slightly bent between the prongs *cd*.

Another device for the same purpose is shown in figs. 146 and 147. This consists of two nearly vertical spindles, slightly inclined towards each other. These are attached to the end of the truck in any suitable manner. Each spindle carries an excentric sleeve, as shown, which can turn freely upon it. When the rope falls between the sleeves, it turns them round until it is gripped between them. In the figures, the thin sides of the sleeves are turned inwards, and the sizes shown are for a  $\frac{3}{4}$ -inch rope. This clip was first introduced in England many years ago.

It is not necessary to give any details of the use of endless ropes hung under the trucks, because this system is distinctly inferior to the overhead rope for surface work, although very valuable for underground haulage in coal mines, and other cases where nearly horizontal seams or reefs are worked.

## CHAPTER XL.

### BELT CONVEYORS, TAILINGS WHEELS AND PUMPS.

THE use of this means of transporting material, both in filling and discharging vats, is likely to be considerably extended in the near future. It is not necessary to give detailed drawings of the apparatus employed for driving and carrying the belts, because it is advisable to purchase conveyors complete from one of the well-known makers, whose patterns represent the accumulated experience of many years, and of many workers in this special department. We shall therefore limit this discussion to the general features of the system.

Fig. 148 shows one of the early methods of carrying the belt. It is obvious that the outer ends of the cones *aa* must have a greater peripheral velocity than the central cylindrical part of the roller. But the edges of the

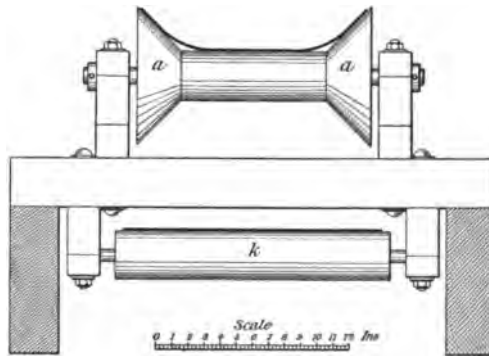


FIG. 148.—Old Mode of Troughing Belt.

belt run at the same speed as its centre, and consequently at some part of its cross-section there must be a continuous slip between roller and belt, which in time destroys the latter.

Fig. 149 illustrates a means of troughing the belt, which overcomes this difficulty, by the use of two inclined rollers *bb*, so that the velocity of roller surface and belt is practically equal.

In an interesting communication to the American Institute of Mining Engineers,\* Thomas Robins jun. describes several modifications in conveyor-

\* Vol. xxvi. p. 78.

belt practice, including the method of troughing the belt illustrated in fig. 150. In this case there is a central horizontal roller *a*, and at each end of it

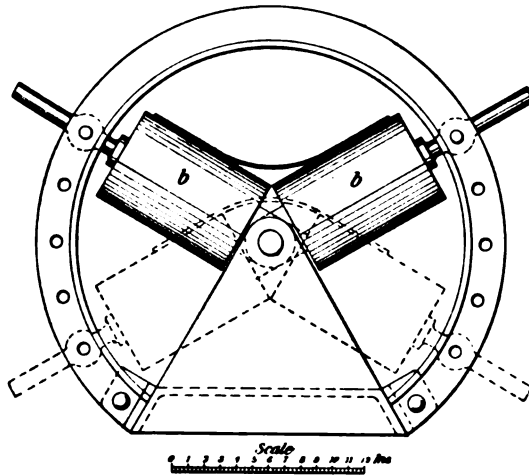


FIG. 149.—Troughing Belt.

a separate roller *d* on an inclined pin *e*, whereby the same advantage is secured as in the arrangement shown in fig. 149. In each of the above cases we have referred only to the rollers that carry the loaded belt, but the part returning empty also wants some support, and this is provided for by rollers, as shown at

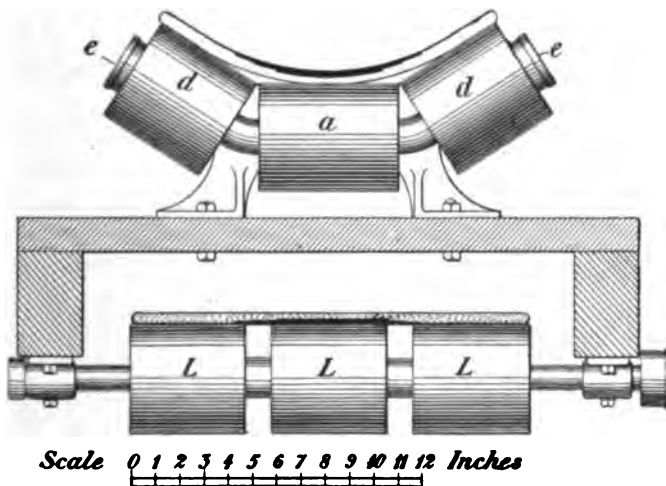


FIG. 150.—Troughing Belt.

K, fig. 148. It is, however, generally preferable to divide these rollers into two or three parts, running loose on fixed shafts, as shown at L, fig. 150. The

best position for the driving pulley is at the delivery end of the belt, because in that case the loaded part of the belt is under the greater tension, while the return portion runs slack.

The following five recommendations of T. Robins jun. may be said to represent the high-water mark of excellence in conveyor-belt practice, as required for filling and discharging vats or for dumping filter-press cakes.

1. The driving pulley should not be less than 30 inches in diameter for small belts (say 18 inches wide), and 48 inches for large belts (say 30 inches wide). The pulleys at each end should be 4 inches wider than the belt.

2. The supporting and troughing rollers should be from 4 to 6 feet apart between centres, and the rollers under the return portion from 8 to 12 feet between centres.

3. The thickness of rubber should not be less than  $\frac{1}{4}$  inch over the actual working width of the belt.

*Note.*—The Robins Belt Coy. and other makers supply belts with thicker rubber in the centre than at the sides. This makers' method is shown in fig. 150.

4. The thickness of canvas should be—

Up to 20 inches wide,	.	.	.	4 ply
„ 24 „	.	.	.	5 „
„ 28 „	.	.	.	6 „
„ 36 „	.	.	.	8 „

5. The material should be delivered to the belt by an inclined shoot, and in the direction of the motion of the belt, with as nearly as possible the same speed.

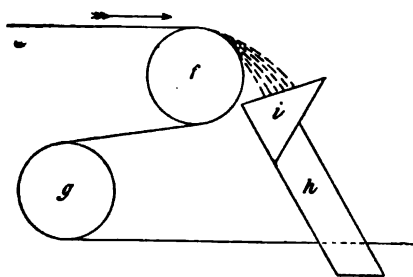


FIG. 151.—Discharging Apparatus.

The ordinary form of apparatus for delivering from the belt at any required position is illustrated diagrammatically in fig. 151.

Two pulleys, *f* and *g*, and a shoot *h*, are fixed in a frame which travels on a track in line with the belt. The loaded part of the belt, which travels in the direction shown by the arrow, passes successively over the pulleys *f* and *g* in the manner shown, so that the material is thrown off

into the hopper *i* at the top of the shoot *h*, which latter projects sideways so as to deliver clear of all the gear.

A shoot may be arranged on either side of the frame, and for large tanks may be made adjustable to deliver at various distances from the belt, and thus lessen the necessity of hand labour for spreading. This frame, called a tripper or discharging carriage, is often made portable and mounted on wheels, so that it may be readily removed from place to place. It is sometimes made to be moved by hand, but is also made so as to be moved by the conveyor

belt. It can be arranged to reverse its direction automatically after a short run, thus moving to and fro, delivering, and distributing its load all the time.

Another means for delivering, which is sometimes useful, is shown by the dotted lines in fig. 149. By moving the rollers into the lower positions there indicated, a convex curvature crosswise is given to the belt, and the material is thereby thrown off on either or both sides.

A special arrangement for spreading the ore in tanks has been devised by H. W. Blaisdell, which distributes the sand centrifugally in the tank. This is being introduced in the U.S.A. by Chas. Butters.

**Capacity of Belts.**—This depends upon the relative bulk and weight of the material to be conveyed. This point will be made quite clear by considering that a belt can carry as much coke as can be piled on to it, the only limit being the liability of falling off during transport, while the same belt would break down under an equal *bulk* of shot. Nevertheless, the belt might easily carry a greater *weight* of shot than of coke in a given time. Therefore all general tables of capacity, whether expressed in weight or bulk, are liable to mislead, unless the material to which they apply is clearly stated. Assuming that the strength of the belt is properly proportioned to its width, the following general formula will apply in all cases by using a suitable coefficient.

Let  $w$  = width of belt in inches

$v$  = speed in feet per minute

$K$  = a constant for each individual material, and

$C$  = the capacity in cubic feet per hour

Then

$$C = Kw^2v$$

For crushed ore or tailings of ordinary specific gravity, the maximum capacity under perfect conditions as to erection and uniformity of loading, we may put  $K = 0.016$ . But it is seldom possible to charge the belt with absolute regularity, and it is therefore safer in practice not to expect more than half this amount. The following table is therefore calculated with  $K = 0.008$ .

TABLE LXXI.—*Approximate Capacity of Troughed Conveyor Belts for Crushed Ore or Tailings, stated in cubic feet delivered per hour.*

Width in inches.	Speed of Belt in feet per minute.						
	200	250	300	350	400	450	500
16	409	512	614	717	819	922	1024
18	528	648	778	817	1057	1166	1296
20	640	800	960	1120	1280	1440	1600
22	774	968	1162	1355	1549	1742	1936
24	921	1152	1382	1613	1843	2074	2304
26	1081	1352	1622	1893	2163	2434	2704
28	1254	1568	1882	2195	2509	2822	3136
30	1440	1800	2160	2520	2880	3240	3600

This is a very conservative basis, and the figures given may be somewhat exceeded in cases where ideal conditions of loading obtain, or even when economy in first cost is of exceptional importance.

*Example.*—Suppose a 500-ton vat at 22 cubic feet per ton has to be filled in 10 hours by a belt running at 350 feet per minute. Then total cubic feet in vat = 11,000, thus requiring 1100 cubic feet per hour. The table shows that a 20-inch belt should be provided.

**Other Conveyors and Elevators for Dry Material.**—There are numerous other mechanical devices which are more or less used for this purpose, such as screw conveyors, push conveyors, and various forms of sheet steel trays attached to ropes or chains. All of these are liable to rapid wear when used for siliceous material, and are therefore less suitable than belts in most cases. It is true that the same objection applies to bucket elevators, which are largely employed in dry crushing mills, but their use is unavoidable, because there is at present no proved efficient substitute for them. There seems to be no reason, however, why internal bucket wheels should not be used for dry material and moderate lifts, provided that they are suitably designed for the purpose. To estimate the capacity of conveyors and elevators which consist of trays or buckets, it is only necessary to know the size of one bucket, their pitch on the chains, and the speed of the apparatus. Allowance has, of course, to be made for irregular supply, and consequent partial filling of some of the buckets.

**Tailings Wheels.**—These are very commonly used on the Witwatersrand goldfield for lifting battery pulp to spitzluten placed above the collecting vats, more particularly in the double-tier plants illustrated in figs. 41 and 42, the only practical alternative in such cases being a tailings pump.

Two kinds of wheel have been used, namely, that with outside buckets like an ordinary water wheel, and the internal bucket type. The disadvantage of the external bucket wheel is that it can only pick up the pulp by dipping into it, and can only discharge by throwing the pulp forward into the upper launder, so that the drip and splash are much greater than with a well designed wheel with internal buckets, into which the pulp can be delivered by a launder, without any splashing whatever. Consequently the latter type only is used at the present time.

The mechanical stresses to which a lifting wheel is subjected are similar in nature and degree to those which occur in a water wheel, so that the rules as to the strength of the various parts laid down in special treatises on the latter may be followed. The chief essential difference is, that in the internal lifting wheel the buckets must laterally overhang their supporting arms in order to allow of the reception and discharge of the pulp.

Double wheels have been built with a set of buckets on each side, and the driving pulley between them, so that they are perfectly balanced laterally.

In the most recent practice the driving pulley is always of large diameter, because by this means the stresses developed in transmitting the power from the pulley to the buckets are confined to a small portion of the framing,

namely, the part near the outer circumference, while the chief function of the radial arms is then only to transfer the weight of the rim of the wheel to the shaft. A belt or rope drive is now generally used.

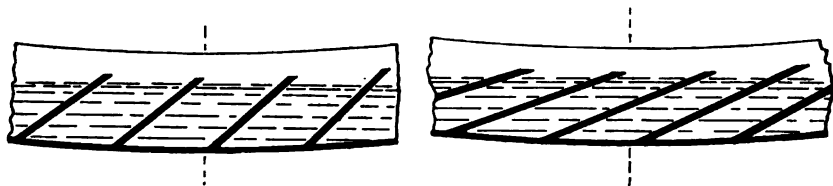
In deciding upon the diameter necessary for a given lift, allowance must be made for the height necessarily lost between the lip of the bucket in its highest position and the bottom of the receiving launder. Also, for the radial depth of the buckets, and for the small space between them and the supply launder.

In practice the ratio of outside diameter to nett lift is generally about 1·3 to 1, so that about 23 per cent. of the power expended is unavoidably wasted from this cause alone.

The buckets are generally formed by plane divisions, and some diversity of opinion has been displayed in existing examples as to the proper angle at which these dividing planes should be placed. It is therefore worth while to briefly examine the conditions which affect this element of the design.

For this purpose the operation of the buckets may be considered in three separate stages, namely, filling, lifting, and discharging.

Figs. 152 and 153 show a few buckets in their lowest position for two different cases, that is to say, in fig. 152 the divisions make a small angle



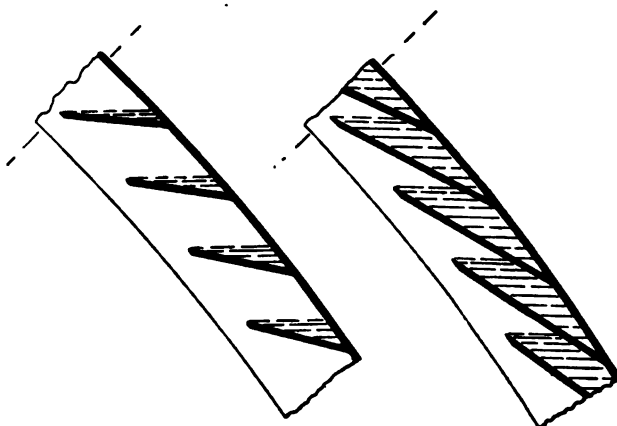
Figs. 152, 153.—Positions of Buckets.

with the radius, and in fig. 153 a large one. It is at once apparent that the mouth of the bucket in the second case is much narrower than that in the first figure. Consequently, if the angle is made very large, the mouth becomes so extremely narrow that the air inside is liable to be trapped by the entering pulp, so that the bucket could not readily be filled, and the amount of splash would be increased. A wheel designed with a very large angle could, therefore, only be run slowly in order to give the buckets time to fill properly.

In figs. 154 and 155 the same buckets are shown at the point where the discharge is intended to begin. In the former figure nearly the whole of the pulp in each bucket has been lost, having dribbled back down the inside of the wheel. To overcome this difficulty, fixed aprons are placed close to the inner edges of the buckets, so that the water cannot so easily escape. In fig. 155, however, a much larger proportion of the pulp has been lifted to the discharge point, so from this point of view the larger angle is far more satisfactory, because there is less drip, and also less power wasted in lifting a portion of the pulp, through part of the height, over and over again.



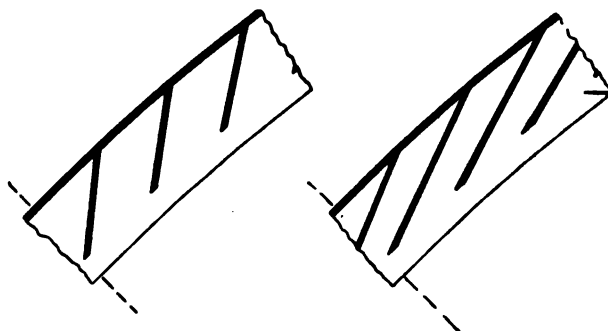
These drawings, of course, are only quite correct for wheels moving very slowly, because a greater speed will cause a greater portion of the liquid to be lifted, owing to its inertia. But in any case the difference between the



FIGS. 154, 155.—Positions of Buckets.

quantities lifted by the two forms of buckets will be very considerable, and it is better to have a slowly moving wheel working efficiently, than a quickly moving one working badly, and with a lot of splash and drip.

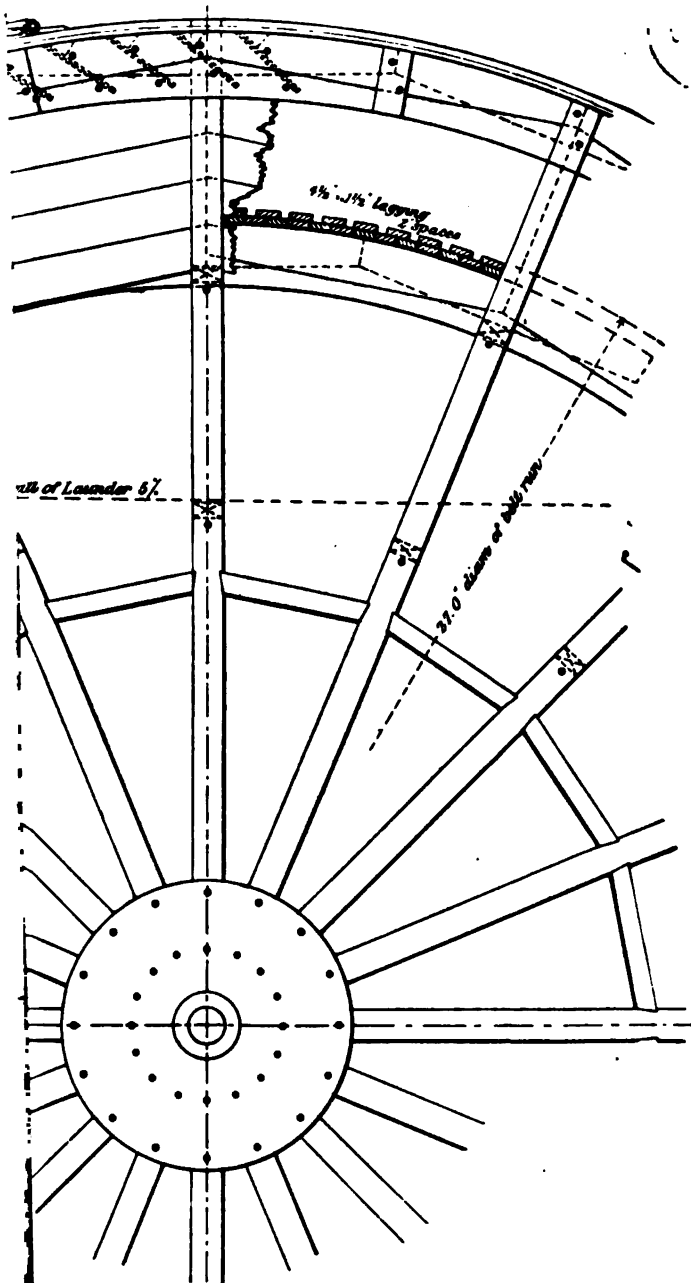
Figs. 156 and 157 show the position of the buckets at the end of the discharge. Here the first named has an apparent advantage, because the



FIGS. 156, 157.—Positions of Buckets.

dividing planes attain a more vertical position. But the discharge is so rapid in either case that this is of no real benefit except in very high wheels, in which the sand tends to settle in the bucket before reaching the point of discharge.

By placing the divisions very close together, as in fig. 158, the total



al Elevation.



quantity lifted can be increased, but only at the risk of some of the sand remaining in the buckets, owing to their mouths being too narrow.

It appears, then, that no absolute rule can be laid down as to the best possible inclination of the bucket divisions, but an angle of  $65^\circ$  may be recommended as avoiding the disadvantages of either extreme. With this angle a suitable pitch for the buckets may be arrived at in the following manner, as shown in fig. 159, where  $ab$  is the outer circle of the wheel,  $cd$  the inside of the shrouding, and the dotted line  $ef$  represents the inner circumference of the buckets. Let  $jk$  represent a radius of the wheel at the point where



FIG. 158.—Position of Bucket.

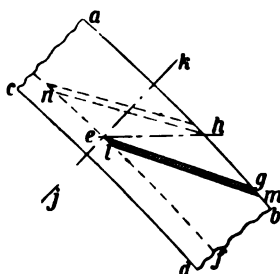


FIG. 159.—Pitch of Buckets.

discharge commences, and which cuts the circle  $ef$  at the point  $e$ . From  $e$  draw a horizontal line cutting  $ab$  at  $h$ , and draw  $eg$  making an angle of  $65^\circ$  with  $jk$ . This latter line is the inside of one bucket. The thickness of the division and of the hard-wood lining (if any) is marked off by the line  $lm$ , and the distance  $hm$  is the required pitch. As this may not be an exact sub-multiple of the circumference, it must be modified accordingly. It will be seen that the area of the triangle  $ehg$  is very nearly equal to half that of the figure  $enhg$ , hence for wheels laid out in this way the capacity per revolution is given with sufficient accuracy by the following formula.

Let  $R$ =radius in feet at back of buckets

$r$ =radius in feet at mouth of same

$n$ =number of buckets

$l$ =length of divisions measured on incline in inches

$t$ =thickness of same in inches

$B$ =inside width of bucket

$C$ =capacity in cube feet per revolution

Then

$$C = B \left( \frac{\pi R^2 - \pi r^2}{2} - \frac{ln}{144} \right)$$

The quantities  $\pi R^2$  and  $\pi r^2$ , being simply the areas of the respective circles, can be at once written down from tables. When the divisions are very thin, as, for instance, those of sheet iron without wooden lining, the term  $\frac{ln}{144}$  may be neglected, and we get

$$C = B \left( \frac{\pi R^2 - \pi r^2}{2} \right)$$

The tailings wheel illustrated in figs. 160 and 161, from drawings kindly lent by Ernest Williams, of Johannesburg, is of the double type, that is, with buckets on both sides of the arms, and with a driving pulley in the middle.

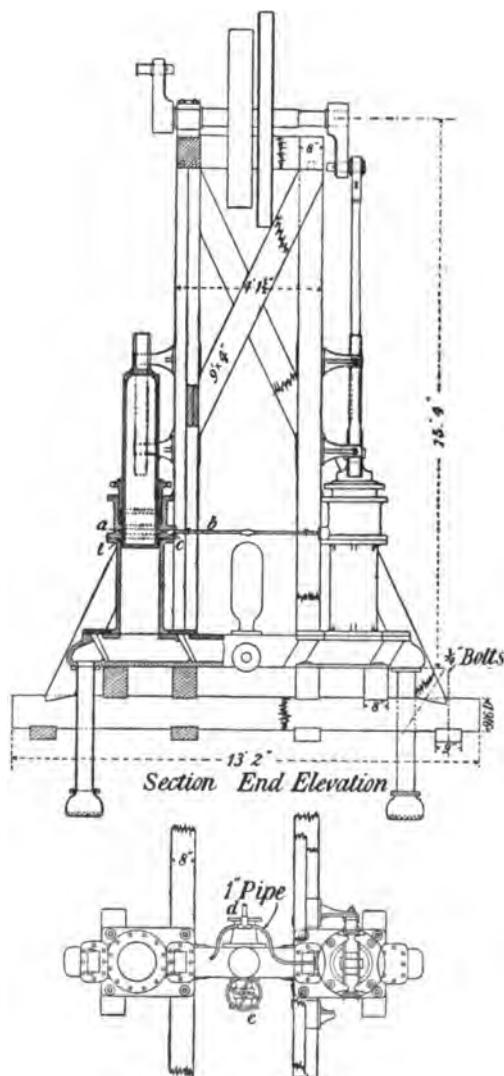


FIG. 162.—Tailings Pump ; End View, etc.

It is 33 feet in diameter, and is designed for a nett lift of 24 feet. When running at 4 revolutions per minute, it is suitable for lifting the pulp from a 100-stamp mill, crushing, say, 500 tons per day. It has 96 buckets on each

side, laid out to a tangent circle 24 ft. 6 in. in diameter, so that they make an angle of about  $50^\circ$  with the radius.

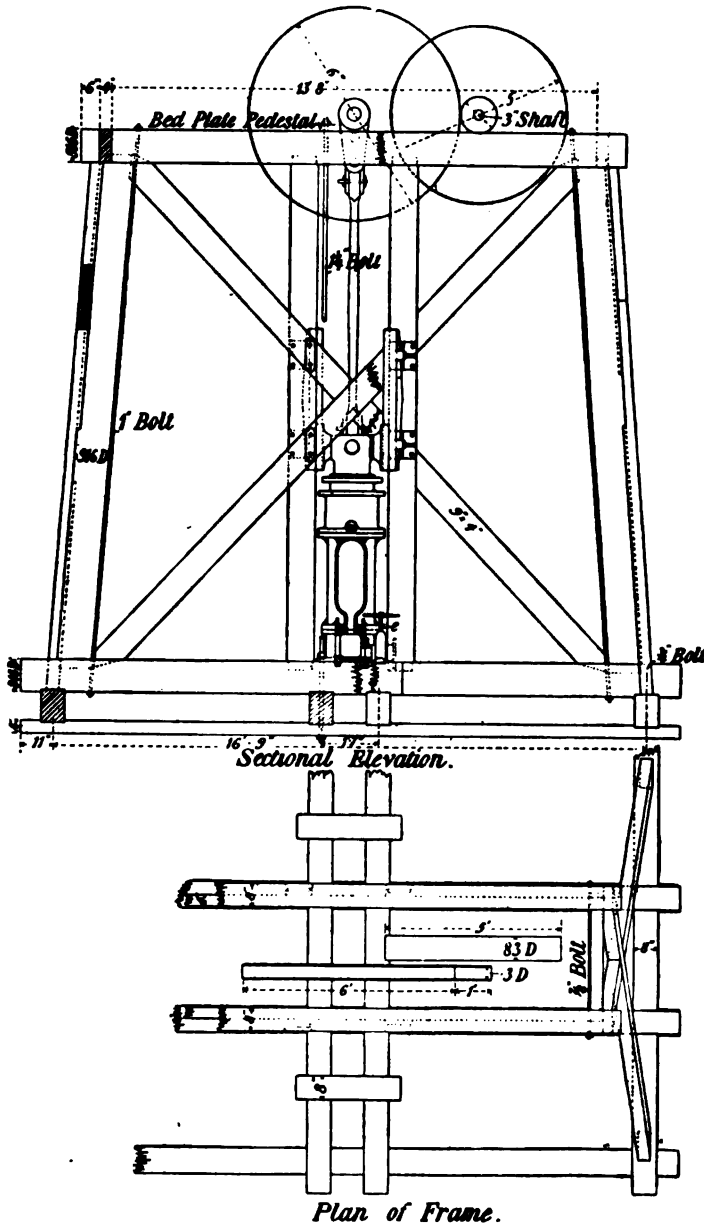


FIG. 163.—Tailings Pump ; Side View of Frame, etc.

**Tailings Pumps.**—At many mines, especially in Australia, double-throw plunger pumps are used for lifting battery pulp, with satisfactory results.

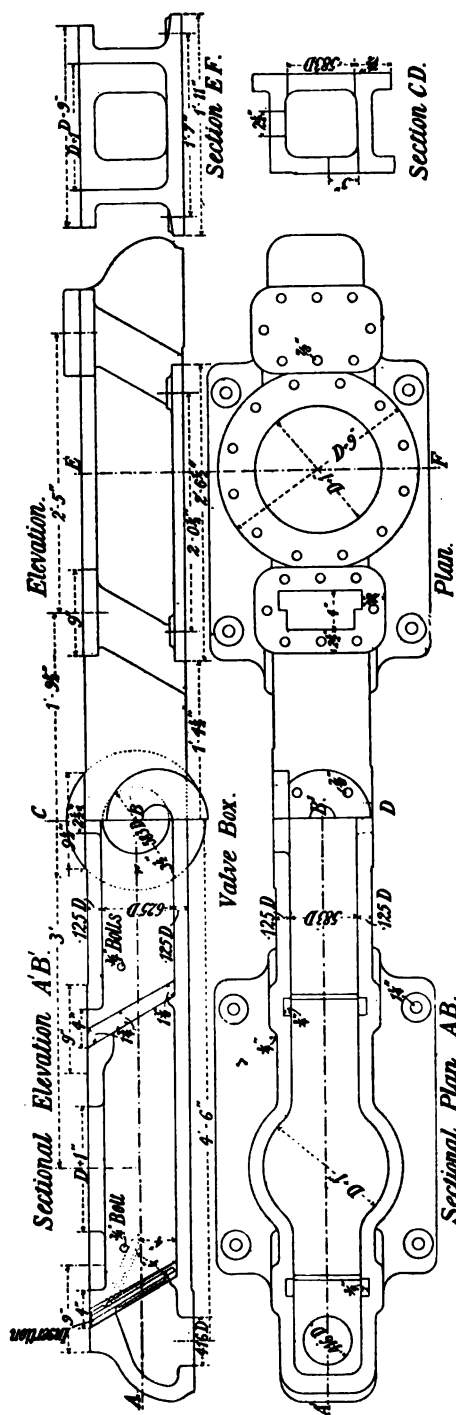
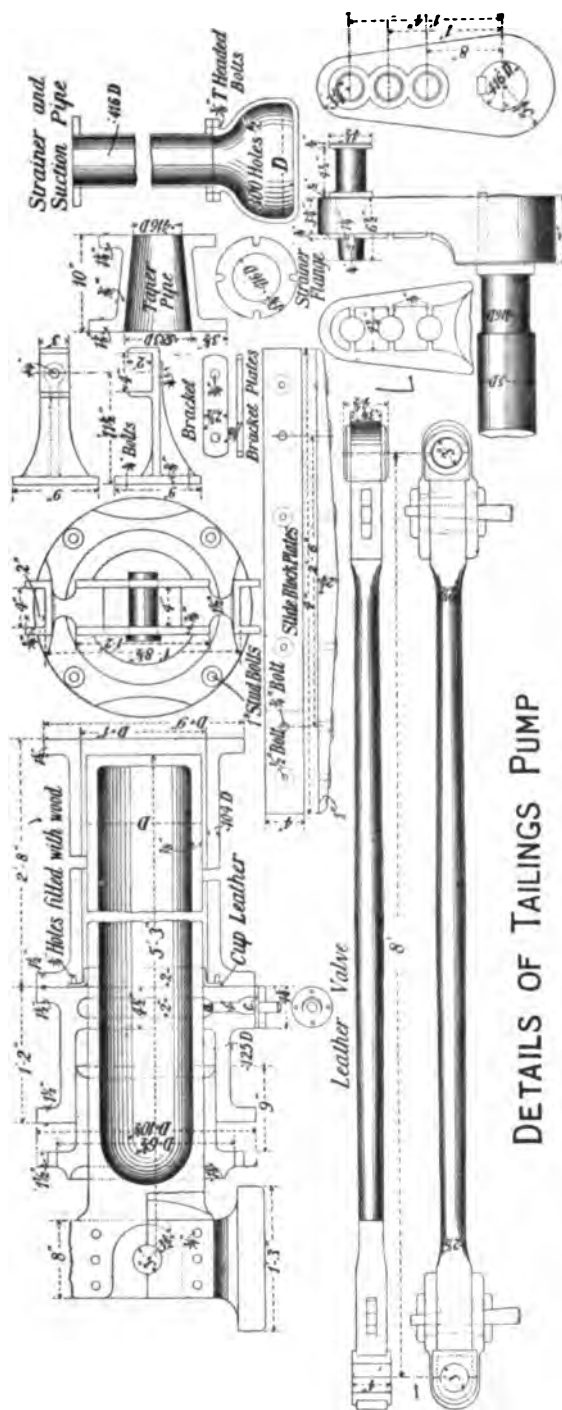


FIG. 164.—Valve Box for Tailings Pump in Detail ; D equals Diameter of Plunger.





Figs. 162 to 165 show a simply designed tailings pump, such as may be made in small engineering works usually found in mining districts. The dimensions given are for cast iron, the spur gear, connecting rods and crank shaft of steel, and the timber frame of hard wood. The lift may be anything up to 150 feet.

The chief difference between tailings pumps and water pumps is that in the former all the passages have to be choked, so as to give a high velocity to the flow of pulp through the passages and pipes, without unduly increasing the speed of the pump; the obvious object being to prevent the sand settling while it is being transferred. Also, it will be noticed in figs. 162 and 165 that there is a half-round groove *a* just below the stuffing-box, which is connected with a pipe *b*, provided with a valve *c*, and below this groove is a cup-leather which fits tightly around the plunger. The pipe *b* is connected with a cistern of clean water fixed at a suitable height, so that the water may flow by gravity. In the up stroke of the plunger the valve *c* opens and admits a supply of clean water to the groove *a*, which then runs down between the plunger and the cup leather, washing off any adhering grit, which is thus prevented from getting to the packing. In the down stroke the valve *c* closes by the pressure from inside, and the cup leather is pressed tightly around the plunger, and thus prevents grit being forced into the groove or packing. The valve box shown is arranged for ordinary clack valves, but some makers prefer to use disc valves, for which a box can be easily designed. An air vessel should be provided to keep the flow more regular, and to relieve sudden internal shocks to which these pumps are liable. The delivery pipe is attached to *d*, and must be in a regular line, free from kinks where sand can lodge. A sluice valve *e* should be provided to empty all the passages immediately the pump stops. A fairly high speed of 15 to 25 strokes a minute is desirable, and the capacity may be regulated, in case of need, by adjusting the crank pin, which is preferable to varying the speed, as a short, quick stroke prevents settling better than a long, slow one. Leather may be used for the valves, but rubber will be found more durable; old rubber belting answers the purpose very well. These last three to four months, and plunger cup leathers last about a year.

It is not desirable to make the suction pipe more than 8 feet long, and the sumps should be small, but large enough for a man to get into and clear. A fall of 2 or 3 feet for the pulp into the sumps prevents settling.

The following examples illustrate the sizes used in practice, from which others can be readily calculated, for any given case. At the Bonanza mine, Johannesburg, for 55 stamps, crushing about 260 tons per day, one double-throw plunger pump, 12 inches diameter by 48-inch stroke, lifts the whole pulp, say 2600 tons, to a height of 45 feet. A second pump is provided as a stand-by, and to allow of repairs being effected without interfering with the work.

At the May Consolidated mine, Johannesburg, one double-throw plunger pump, 15 inches diameter, lifts the pulp from 100 stamps to a height of 65 feet. These two examples give a mean figure of 1 circular inch of plunger area for each 20 tons of pulp to be lifted per day.

## CHAPTER XLI.

## DESIGN AND CONSTRUCTION OF SPITZLUTTE AND SPITZKASTEN.

THE theory of bodies falling in water has been worked out by Rittinger, the inventor of the spitzkasten, and published in his *Lehrbuch der Aufbereitungskunde* from 1867 to 1873, and the reader is referred to that work for full details. Only a brief summary of his results can be given here, for the use of those to whom the original work is inaccessible.

Let  $W$  = pressure exerted on a flat horizontal surface by a vertically flowing stream of water whose velocity is  $v$ . Let  $f$  be the area of the surface.

## Then

$$W = 51fv^3 \quad (1)$$

If  $f=1$  square metre, the pressure is  $w=51v^2$  kilos.

If the velocity  $v = 1$  metre per second, the pressure on one square metre is  $a = 51$  kilos., and the equation may be written thus—

$$w = av^2 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (2)$$

Whether a body will fall in the stream depends not only upon the weight of the body, but also upon the size and form of the surface opposed to the stream.

Rittinger gives the following equations for the pressure on variously shaped bodies:

A wedge pointing downwards when  $h$  = height and  $b$  = breadth.

$$\alpha_1 = \frac{\alpha}{\frac{4h^2}{h^2} + 1} \quad . \quad . \quad . \quad . \quad . \quad . \quad (3)$$

If the section of the wedge is an equilateral triangle, then  $\alpha_1 = \frac{\alpha}{4}$ . And if the angle at the bottom of the wedge is a right angle, then  $\alpha_1 = \frac{\alpha}{9}$ .

The actual meaning of the above results is, that while a cube opposing a flat face to a stream of unit velocity will lose 51 kilogrammes of its weight for one square metre of surface, a right-angled wedge will lose only one-half and an equilateral wedge one-quarter of this amount, on account of the upward motion of the water.

**For a cone when  $\beta$  = half the angle at the apex,**

$$a_2 = a \sin^2 \beta$$

**Therefore when**

$$\beta = 45^\circ \quad a_2 = \frac{a}{2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (4)$$

**And for a hemispherical surface,**

$$g_3 = \frac{a}{2} \quad . \quad . \quad . \quad . \quad . \quad . \quad (5)$$

So that in all cases, whether the body is a wedge, cone, or hemisphere, when the depth is half the breadth, we have the same result; that is to say,

$$a_1 = a_2 = a_3 = \frac{a}{2}$$

which for unit area and unit velocity = 25.5 kilogrammes.

Rittinger, as the result of numerous experiments with spherical bodies from 1 to 10 millimetres diameter, and of specific gravities from 1.3 to 11, gives the value of  $a_3$  at 25.5 kilos very nearly, so that within these limits calculation and experiment agree very closely.

The above equations have been quoted, because they lead to the general conclusion, that the resistance of the water to a falling body depends more upon the relative depth and width of it than upon the actual shape of its specific lower surface.

General formulæ for bodies falling in water are given as follows—

$s$  = distance through which the body falls

$t$  = time in seconds at the end of which the motion of the body is considered

$d$  = diameter of sphere

$\delta$  = specific gravity

$\gamma$  = 1000 kilos = weight of 1 cubic metre of water

$$C = \sqrt{\frac{2\gamma}{3a_3}}$$

Then

$$s = 5.11t\sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (6)$$

And from this the mean velocity

$$v = \frac{s}{t} = 5.11\sqrt{d(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (7)$$

The following corollaries of the last equation are of importance.

(1) With two equal falling bodies of different diameters  $d_1$  and  $d_2$  and different specific gravities  $\delta_1$  and  $\delta_2$ , then

$$\frac{d_1}{d_2} = \frac{\delta_2 - 1}{\delta_1 - 1} \quad . \quad . \quad . \quad . \quad . \quad (8)$$

That is to say, if the equal falling bodies have different densities, their diameters are also different, and *vice versa*, the denser body having the smaller diameter, and consequently the smaller volume. For quartz and ordinary iron pyrites, the diameters are as 2.5 to 1 nearly, while the volumes are as the cubes of these numbers, namely, about 16 to 1.

(2) If  $P_1$  and  $P_2$  represent the absolute weights of two equal falling bodies, they are thus related—

$$\frac{P_1}{P_2} = \frac{\delta_1(\delta_2 - 1)^3}{\delta_2(\delta_1 - 1)^3} \quad . \quad . \quad . \quad . \quad . \quad (9)$$

Thus, for quartz and pyrites the relative weights are about 10 to 1 as calculated from this last equation, and in all cases the denser body has not only a smaller diameter and volume, but also a smaller absolute weight.

These last considerations show clearly why it is possible to take out a large proportion of the pyrites from battery pulp, together with only the coarsest particles of the quartz, in a spitzlutte, as now so largely practised on the Rand, although, as we shall see later, the actual numerical results of the last two equations do not apply to very small grains of irregular shape, as shown by the experiments of R. H. Richards quoted on p. 335.

So far as we have yet gone, Rittinger's investigations deal only with bodies of regular shape. In treating of irregular bodies, he designates the resistance of the water by

$$\alpha_4 = \zeta \alpha = \zeta 51 \quad . \quad . \quad . \quad . \quad . \quad . \quad . \quad (10)$$

where  $\zeta$  is a coefficient, depending upon the form of the grains, which is determined experimentally.

Experiments with grains, selected according to shape as well as according to size, gave the following figures—

Rounded grains	$\zeta=1.25$	$a_4=65$
Longish	$\zeta=1.86$	$a_4=90$
Flat	$\zeta=2.35$	$a_4=120$

When the weight and sp. gr. of a grain is known, the diameter of a sphere of equal weight can be calculated. This 'ideal diameter' of the grain may be represented by  $d_1$ . Then the velocity of the falling grain is

$$v = \sqrt{\frac{2\gamma}{3a_1} d_1(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (11)$$

### Putting the constant factor

$$\sqrt{\frac{2\gamma}{3a_1}} = C_2$$

**we get**

$$v = C_2 \sqrt{d_1(\delta_1 - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (12)$$

Assuming 50 per cent. of rounded grains and 25 per cent. of each of the other kinds, the average value of  $C_2 = 2.85$  as compared with  $C_1 = 5.11$  for perfect spheres.

But Rittinger goes a step further in a practical direction, for instead of using this ideal diameter, he connects it with the size of sieve holes  $D$  by putting  $\mu D$  for  $d_l$  in the equation for  $v$ , thus—

$$v = \sqrt{\frac{2\gamma}{3\alpha_1} \mu D(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad . \quad (13)$$

**Then putting the constant quantity**

$$\sqrt{\frac{2\gamma\mu}{3\alpha_1}} = C_3$$

he finally obtains the equation

$$v = C_1 \sqrt{D(\delta - 1)} \quad (14)$$

and from experiments deduces the following values—

Rounded grains	$\mu=0.73$	$C_3=2.73$
Longish	$\mu=0.80$	$C_3=2.37$
Flat	$\mu=0.67$	$C_3=1.92$

Average value, on the same assumption as before, as to the relative quantities of each shape,  $C_3 = 2.34$ .

There is a slight discrepancy between the values given for  $\mu$  and  $C_3$  in the case of longish grains, but as it only affects the second place of decimals in the average value of  $C_3$  it is unimportant.

The average velocity for irregular grains having specific gravity  $\delta$ , and of the sieve class produced with holes of diameter  $D$ , can therefore be found from the following empirical equation—

$$v = 2.34\sqrt{D(\delta - 1)} \quad . \quad . \quad . \quad . \quad . \quad (15)$$

Now, if the grains are placed in an upward current of water whose velocity is equal to the mean velocity of the falling grains, a state of equilibrium is produced for this particular kind of grain, and all grains which fall more slowly will be carried upward and thereby separated from those which fall more quickly, as already explained with reference to figures 38 and 40, on pages 177 and 178.

Rittinger sums up this part of his work in the following words:—

“The empirical values of  $C_3$  are nevertheless diverse, even with bodies of like forms; one must therefore despair of an exact formula for irregular bodies in water, and must be contented with the above approximation (*schliesslichen*), which is quite sufficient for practical work.”

He gives some tables calculated from the last equation, from which we have compiled the following figures. We have also given the diameters in inches and the velocities in feet per second.

TABLE LXXII.—*Velocity required in upward flowing stream of water to hold irregular bodies in equilibrium.*

Sieve diameter in inches.	Velocity $v$ in feet per second.			Sieve diameter in mm. = $D$ .	Velocity $v$ in metres per second.		
	Galena sp. gr. = 7.5.	Pyrites sp. gr. = 5.	Quartz sp. gr. = 2.6.		Galena sp. gr. = 7.5.	Pyrites sp. gr. = 5.	Quartz sp. gr. = 2.6.
0.157	1.286	0.984	0.640	4	0.392	0.30	0.195
0.118	1.115	0.853	0.558	3	0.340	0.26	0.170
0.110	1.073	...	0.528	2.8	0.327	...	0.161
0.079	0.912	0.722	0.449	2	0.278	0.22	0.137
0.055	0.761	...	0.377	1.4	0.232	...	0.115
0.039	0.640	0.492	0.318	1	0.195	0.15	0.097
0.028	0.541	...	0.269	0.71	0.165	...	0.082
0.020	0.453	0.328	0.239	0.5	0.138	0.10	0.073
0.014	0.380	...	0.190	0.35	0.116	...	0.058
0.010	0.321	...	0.157	0.25	0.098	...	0.048
0.005	0.226	...	0.112	0.125	0.069	...	0.034

The next table shows the results obtained by R. H. Richards,\* of Boston,

\* Spitzkasten and Slime Tables, *Proc. Am. Inst. of Min. Eng.*, vol. xxvii. p. 76.

U.S.A., from an extensive series of experiments. As noted at the foot of this page, the diameters were obtained by measurement of sample grains, and not by the size of sieves as in Rittinger's experiments.

A comparison of these figures with those of Table LXXII. shows a certain amount of disagreement, and to make clear the nature and extent of this, we

TABLE LXXIII.—*Diameters of Quartz and Galena particles which are equal-settling in upward currents specified under free settling conditions.*

Particles fall in Current of inches per second.	Particles rise in Current of inches per second.	Diameter of Particles in inches.		Ratio between diam. of Particles actually obtained.	Diameter of Particles in millimetres.		Particles fall in Current of milli- metres per second.	Particles rise in Current of milli- metres per second.	Ratio between diam. of Particles averaged by curve.
		Quartz.	Galena.		Quartz.	Galena.			
·000	·050	·00119*	·00076*	1·55	·0301*	·0194*	0·00	1·26	1·54
·050	·099	·00132	·00078	1·69	·0335	·0198	1·26	2·51	1·68
·099	·199	·00224	·00115	1·95	·0568	·0292	2·51	5·05	1·82
·199	·292	·00304	·00162	1·87	·0772	·0412	5·05	7·42	1·96
·292	·394	·00387	·00192	2·01	·0982	·0488	7·42	10·01	2·09
·394	·577	·00561	·00242	2·32	·1423	·0613	10·01	14·68	2·23
·577	·780	·00739	·00284	2·60	·1875	·0721	14·68	19·80	2·35
·780	1·186	·0089	·0041	2·18	·2254	·1032	19·80	30·12	2·48
1·186	1·589	·0135	·0051	2·62	·3416	·1305	30·12	40·37	2·61
1·589	1·972	·0153	·0055	2·76	·3880	·1404	40·37	50·08	2·72
1·972	2·366	·0206	·0067	3·07	·5241	·1708	50·08	60·09	2·82
2·366	2·769	·0232	·0079	2·95	·5892	·1997	60·09	70·34	2·92
2·769	3·160	·0260	·0094	2·77	·6590	·2381	70·34	80·28	3·03
3·160	3·552	·0339	·0108	3·13	·8604	·2750	80·28	90·21	3·12
3·552	3·919	·0403	·0135	2·99	1·0234	·3428	90·21	99·54	3·21
3·919	4·334	·0450	·0138	3·26	1·1424	·3504	99·54	110·09	3·29
4·334	4·726	·0521	·0144	3·62	1·3216	·3648	110·09	120·03	3·36
4·726	5·135	·0560	·0149	3·77	1·4224	·3776	120·03	130·43	3·42
5·135	5·526	·0562	·0166	3·39	1·4256	·4208	130·43	140·37	3·49
5·526	5·918	·0632	·0180	3·52	1·6032	·4560	140·37	150·31	3·54
5·918	6·303	·0664	·0181	3·67	1·6848	·4592	150·31	160·09	3·59
6·303	6·691	·0689	·0182	3·78	1·7488	·4624	160·09	169·95	3·63
6·691	7·106	·0710	·0207	3·44	1·8032	·5248	169·95	180·51	3·66
7·106	7·826	·0778	·0228	3·42	1·9744	·5776	180·51	198·78	3·70

give in Table LXXIV., in parallel columns, a selection of figures from the two series. As Rittinger's velocities are for equilibrium, we have taken the mean of Richards' pair of velocities in each case as fairly representing the same condition.

The first point to be noted is that for quartz grains of 1 mm. diameter, the two figures agree very closely, as there is less than 5 per cent. difference between them; and for galena grains of 0·35 mm. diameter, the difference is only 10½ per cent., reckoned on the smaller figure. It is probable, therefore,

\* The length and width of each of ten grains of quartz and of galena were measured for each product by microscope micrometer, and the mean of each 20 measurements is called the average diameter.

that the experiments from which Rittinger's coefficients were derived were carried out with grains of about these sizes. For both materials, however, Richards' results give a much greater range of velocities for the same range of diameters. In other words, Richards' figures for large grains show a greater velocity than Rittinger's, and for small grains a less velocity. It is clear, therefore, that there is some force in operation which increases in relative importance as the particles decrease in size, and which acts so as to retard the fall of the grains, and thereby aids the upward current, and it appears likely that this is due partly to the skin friction of the grains themselves. For this friction will tend to cause the grain to move with the water, because it opposes any relative motion between the two. Also the friction varies as the square of the diameter, while the weight varies as the cube thereof, so that the frictional resistance becomes relatively greater for small grains than for large ones. In Rittinger's investigation, skin friction was not taken into account, and it therefore requires modification to allow for this retardation of the fall of the

TABLE LXXIV.—*Comparison of Rittinger's and Richards' Results.*

Diameter of Particle in millimetres.	Velocities for Quartz in metres per second.		Velocities for Galena in metres per second.	
	Rittinger.	Richards.	Rittinger.	Richards.
2	0·137	0·192	...	...
1	0·097	0·093	...	...
0·5	0·073	0·053	0·138	0·167
0·35	0·058	0·036	0·116	0·105
0·25	0·048	0·028	0·098	0·077
0·125	0·034	0·011	0·069	0·034

grain. Pending further investigation in this direction, his equation is, however, very useful for application in cases where experimental data are not available, and this comparative examination of it, in the light of Richards' results, serves to show in which direction it is likely to err in any particular case.

The column of ratios between diameters of quartz and galena grains in Table LXXIII. shows that equation 8 on p. 332 does not hold good numerically for very small grains. Thus the equation gives the relative diameters of equal-falling grains of quartz and galena as 4 to 1, while in the table the ratio varies from 3·78 to 1·55. In spite of certain irregularities which are unavoidable in such work, the gradual decrease in the ratio as the particles become smaller is very marked, and, on the whole, very uniform. This point should also be borne in mind when using equation 15 for determining the size of spitzlutte. This discrepancy between calculation and experiment is also probably due, to some extent, to the skin friction.

It would appear from the foregoing that gravity and skin friction were the only factors of importance that exert influence on bodies falling through water. This is the case with particles of an appreciable size; but as the particles become finer, other factors, viz., viscosity of the water and electro-static repulsion of the particles (see Chap. XXIX.), become of even greater importance.

It is to be observed, however, that the principles elucidated by Rittinger's investigation are untouched by these discrepancies, and that a rising stream of water, as employed in a spitzlutte, is a suitable and fairly satisfactory means for (1) separating grains of equal density according to their size; (2) for separating equal-sized grains according to their specific gravity; and (3) for separating, from mixed bodies of different sizes and densities, a group of equal-falling grains. As previously mentioned, in cyanide work the last-named operation is usually performed by the first spitzlutte, after which the function of the other boxes is principally to size the remaining grains.

The practice, somewhat recently introduced, of speaking of pointed boxes as 'sizers,' is therefore inaccurate; they are distinctly hydraulic classifiers, capable of several kinds of classification. Rittinger used the word 'class' to designate a group of grains of equal size, and the word 'sort' for a group of equal-falling bodies, but in English, as the latter word has a quite different and well-established meaning in metallurgy, it is not advisable to use it for this purpose.

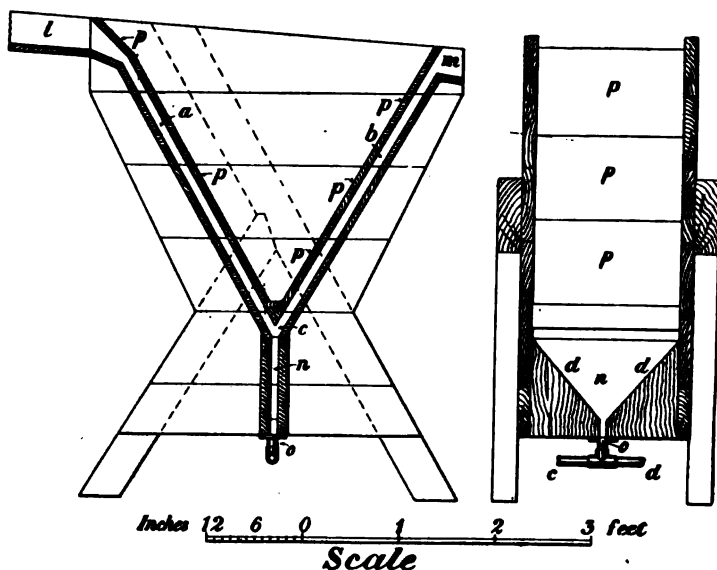
However, as far as the cyanide process is concerned, the chief value of pointed boxes lies in their capability of separating coarsely crushed particles from the finer ore. They are imperfect for this object, but are the best appliances we have for the work. The coarse particles fall through the rising stream of water and separate largely, according to their specific gravities, while the fine particles are prevented from falling, chiefly by the viscosity of the water or pulp, the skin friction and electro-static repulsion. If gravity was the only factor acting, we should be able to separate all particles according to their specific gravities, and pointed boxes would become concentrators, since it is the heavy particles that chiefly carry the gold, but viscosity and the other factors always exert their influences, and prevent the finer particles from falling, no matter what their specific gravity is. Thus a set of conditions that would be applicable to separate coarse particles, according to size, when gravity is the main factor acting, would not be applicable to fine particles, for then viscosity of the water and electro-static repulsion would be the main factors.

In the cyanide process we want first to separate the coarse particles, and finally to separate the very fine particles or slime; and it is clear that an appliance constructed on principles that would be perfect for the former case, would be imperfect for the latter, and *vice versa*. For this reason chiefly, in practice, equally good results are often obtained with apparently crude appliances, as with those constructed on recognised theoretical principles.

**Practical Design of Spitzlutte.**—For a good separation, it is necessary that the rising stream should have, as nearly as possible, the same velocity at



every part of its cross section. This desideratum is attained by passing it through a rectangular passage, one of whose dimensions is very small relatively to the other. Thus, in figs. 166 and 167 the passages *a* and *b* are 18 inches by 1½ inch. In those figures the pulp arriving by the launder *l* passes down passage *a* and then upwards through *b* to the overflow at *m*. The selected grains therefore fall out of the flowing stream at *c*, and below this point the space *n*, with inclined sides *d d*, is provided to lead the separated product to the central discharge pipe *o*. Clean water is supplied through pipe *c* at sufficient pressure to keep back the battery water and the finer solids, and so to ensure a clean product. In a series of apparatus of this kind, the second one would have a passage of larger sectional area, in order to obtain less velocity, and consequently a finer product.



FIGS. 166, 167.—Spitzlutte.

From the data in the above tables it is now easy to find the proper dimensions of the passage for a given quantity of pulp, in order to separate grains of a certain size of any substance whose specific gravity is known.

As an example, assume that a spitzlutte is required to separate from a battery pulp the coarsest particles of quartz which have passed through a 25-mesh screen, with holes of, say, 0.026 inch, and that 250 tons of ore are crushed daily. At 9 tons of water per ton of ore, the total quantity of pulp is 74,575 cubic feet per day, or 0.863 cubic foot per second. From Table LXXIII. the necessary velocity is 0.231 foot per second. Then the required section area is equal to

$$\frac{\text{quantity}}{\text{velocity}} = \frac{0.863}{0.231} = 3.74 \text{ square feet}$$

or 538 square inches. Therefore taking, say, 4 inches as the small dimension of the passage, its width must be 11 ft. 2½ ins. This width would require a great depth for the triangular space ( $n$  in fig. 167) below the passage, and it is therefore more convenient to divide it between, say, three boxes, each of which will then be 3 ft. 9 ins. wide. Of course, with the quartz grains of the selected size, the product will also contain smaller particles of all the heavier minerals present, and generally, therefore, a much greater percentage of mineral than the original pulp.

The next step is to determine the difference between the levels of inflow and outflow, which is required to cause the pulp to actually pass through the passage at the velocity for which the spitzlutte is designed. The ordinary formula  $h = \frac{v^2}{2g}$  may be used, and extra head allowed for friction in the passage,

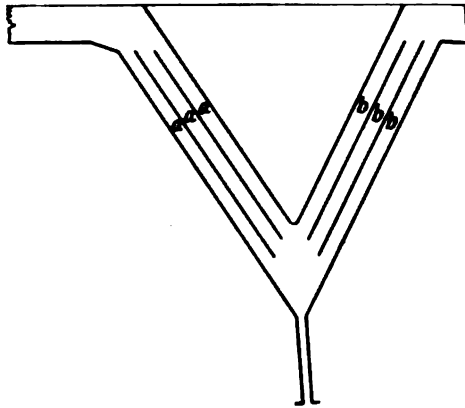
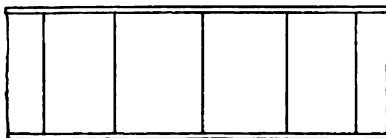
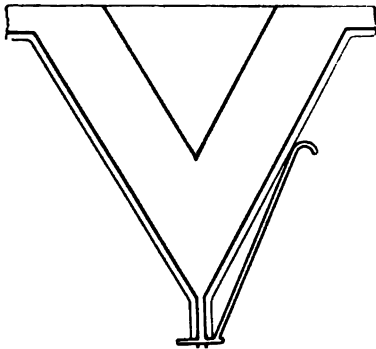


FIG. 168.—Spitzlutte.



FIGS. 169, 170.—Spitzlutte.

but in practically all cases the calculated head is so small, say  $\frac{1}{100}$  inch or thereabout, that it is negligible in comparison with that required to overcome the friction, which may be from  $\frac{1}{2}$  to 2 inches, according to the size of the apparatus and the material of which it is built.

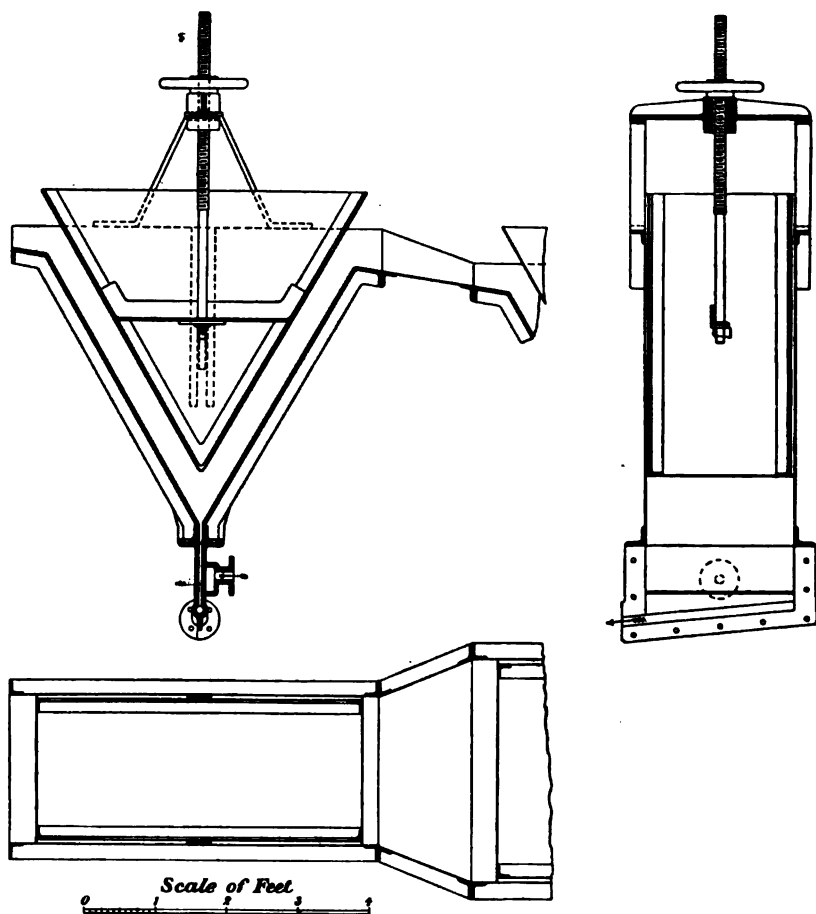
As already mentioned, the object of making one dimension of the passage very small is to secure a uniform product, but the above example shows that this requires sometimes a very great width when the apparatus is constructed with a single passage, as in figs. 166 and 167.

Fig. 168 shows diagrammatically how this objection may be obviated by the use of several parallel passages,  $a a$  and  $b b$ .

But it often happens that extreme accuracy of grain selection is not required, and in such cases the cross-section of the spitzlutte passage may

approximate to a square. Thus, in the above example, by making the smaller dimension 18 inches, the width is reduced to 2 ft. 6 ins. for the whole of the pulp, and the depth is also thereby sensibly diminished. In this case we have an apparatus of the general form shown in figs. 169 and 170.

Figs. 171, 172, and 173 illustrate a neat form of spitzlutte introduced by the United Engineering Coy. of Johannesburg, to whom we are indebted for



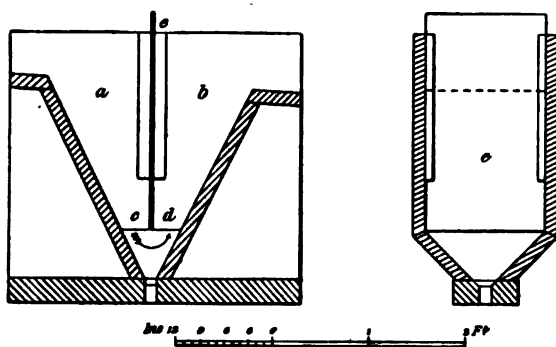
FIGS. 171, 172, 173.—Spitzlutte.

the drawings. The cross-section of the passage can be adjusted to give any required product within limits, by raising or lowering the inner V by means of the screw and nut shown. The pressure water is admitted through a narrow slit which extends right across the width of the apparatus.

Another stage in the modification of the spitzlutte is shown in figs. 174 and 175. In this form the inner V is replaced by a vertical partition *e*, arranged to slide up or down to effect a rough regulation of the product.

This form involves a serious departure from theoretical perfection of product, because the velocity of the stream will not only be different at various portions of any cross-section, but will also vary at different parts of the length of the passage, being slower at *a* and *b* than it is at *c* and *d*. In fact, this last form is really intermediate between the spitzlutte and the spitzkasten, for as the partition *e* is raised, the definite control of the flow is gradually diminished, until, with the entire removal of the partition, the transition of the apparatus to an actual spitzkasten is complete.

**Spitzkasten.**—This form of apparatus does not lend itself so readily to calculation as does the spitzlutte, and the rules for determining its shape and size are chiefly founded upon the results of experience.



FIGS. 174, 175.—Spitzlutte.

Rittinger recommends the use of four boxes in succession, the width of each being twice that of the preceding one. For the first, his rule is to allow a width of one-tenth of a foot for each cubic foot of pulp flowing per minute = 6 feet for each cubic foot per second.

Taking, for example, as before, a battery crushing 250 tons per day = 0.863 cubic foot per second, the widths of the four boxes are 5 ft. 2 ins., 10 ft. 4 ins., 20 ft. 8 ins., and 41 ft. 4 ins.

Linkenbach, in 1887,\* gave the following rule. The width of each box is to be  $1\frac{1}{2}$  times that of the preceding one, and for the first box an allowance of 0.067 foot is to be made for each cubic foot per minute = 4 feet for each cubic foot of pulp per second. These sizes are said to be suitable for material from a  $1\frac{1}{2}$  mm. screen = 0.059 inch, when three boxes are used in series for sand. He further recommends the use of three boxes for slime, of which the first is to have a width of 0.213 foot for each cubic foot per minute, or 12.78 feet for each cubic foot per second. The width of the last box in our example will, by this rule, be 24 ft. 10 ins. This, he says, gives almost clear water overflow. There is a considerable difference in these two sets of figures, due

\* *Die Aufbereitung der Erze*, Berlin, 1887.

probably to the different conditions as regards the material treated and the water used. The presence of acid in the water, or of carbonate of lime in the ore, would facilitate the settlement of the finer particles.

In compiling the following table, we have carefully compared the figures of both writers, as to the sizes of the grains for which the boxes were intended, with the practice on cyanide plants, so far as reliable figures are available.

TABLE LXXV.—*Width of Spitzkasten for separating Grains of Quartz.*

Size of Grain required in the Product.	Width in feet for each cubic foot of Pulp per minute.
Coarse grit, . . . . .	0·05
Sand passing sieve with holes = 0·05 inch, . . .	0·10
„ 30-mesh sieve, . . . . .	0·20
„ 40- to 60-mesh, . . . . .	0·30
„ 90- to 120- „, . . . . .	0·40
Slime from Rand blanket with lime, . . . . .	0·55
Slime settlement without lime, from . . . . .	0·60
„ „ „ to . . . . .	2·50

The length of the box is of less importance. Rittinger recommends 6, 9, 12, and 15 feet for his four boxes, while Linkenbach, for six boxes, gives the following sizes: 1 ft. 8 ins., 2 ft. 6 ins., 3 ft. 9 ins., 12 ft., 16 ft., and 20 ft.

**Inclination of Walls.**—If the walls are not sufficiently steep, the falling grains collect in heaps, especially in the angles of the inverted pyramids, and fall occasionally in masses upon the outlets, which thereby become blocked. Experience shows that an angle of  $55^\circ$  with the horizontal is desirable, and only in extreme cases of limited fall should the absolute minimum of  $45^\circ$  be allowed, when the walls should be lined with glass or other smooth material. The depth inside will therefore, in ordinary cases, be equal to  $\frac{B}{2} \tan 50^\circ = 0·6B$ , where B is the greater longitudinal inside dimension.

**Discharge Orifice and Pressure Water.**—The discharge, when no pressure water is used, is proportional to the square root of the depth, and therefore to  $\sqrt{0·6B}$ . But the quantity of pulp passed through the box, and therefore also the amount of solid matter discharged of the selected size, is proportional to the width. Consequently, with an orifice of given size, a large box should give a thicker product than a small one. The minimum size for the discharge necessary to obviate choking is about  $\frac{3}{4}$  inch diameter.

The sketch given in fig. 176, which is a form sometimes used, illustrates very well the functions of the pressure water.

A is the lower part of the spitzkasten, B is a separate chamber, C the pipe supplying pressure water, and D the outlet for the separated product. We

may assume an ideal case where the pressure in B exactly balances the down-

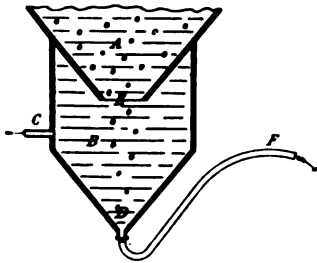


FIG. 176.—Discharge orifice.

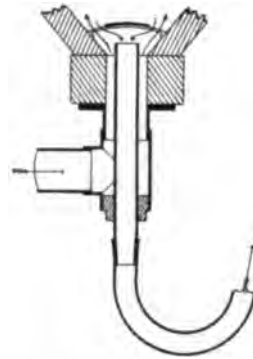
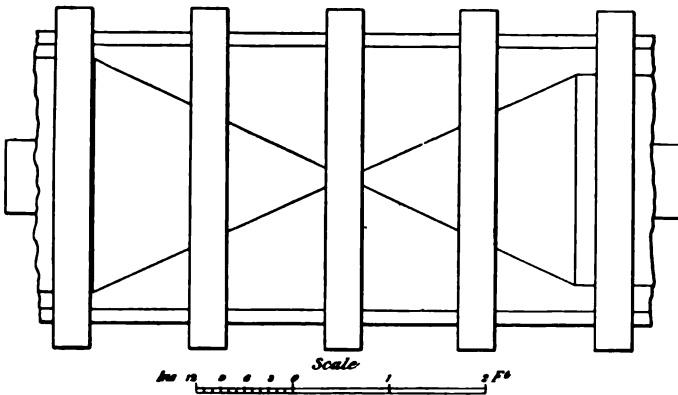
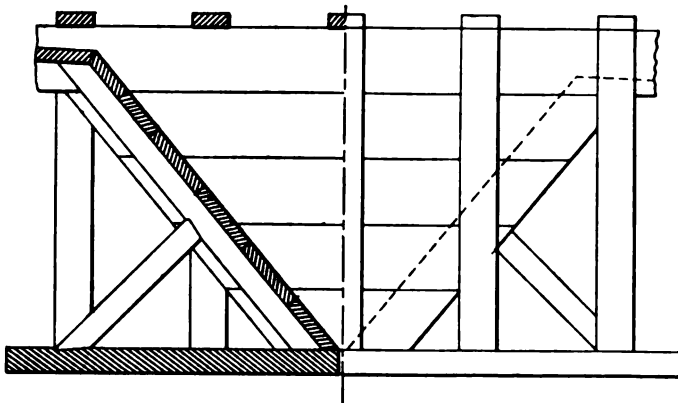


FIG. 177.—Discharge orifice.



FIGS. 178, 179.—Spitzkasten.

ward pressure at the opening E. Then the selected grains would (theoretically) fall from the muddy water in A through E into the clear water in B, and

so through the outlet D, by which only clean water and selected grains would pass. To ensure this last condition in practice, the pressure in B has to be rather more, say 6 to 12 inches, than that in A, and to avoid an excessive waste of clean water a counterpressure has to be set up at D. This is done by attaching a bent pipe F, whose outlet is at a higher level. By using a flexible hose for this purpose, the outflow can be immediately regulated at any

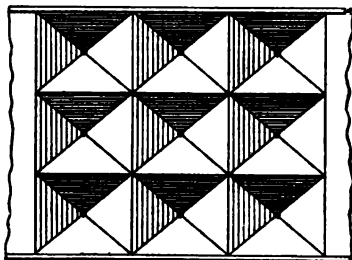


FIG. 180.—Large spitzkasten in plan.

time by raising or lowering the upper end of it. For coarse material, the nett difference between the overflow level and the underflow discharge may be from 3 to  $3\frac{1}{2}$  feet, and for slime from 2 to  $2\frac{1}{2}$  feet.

Fig. 177 shows a discharge pipe arrangement that has given good results in practice, and that shown in figs. 171 and 172 is also very good.

With care, the outflow may be made to contain as much as 37 per cent. of solid matter from the first boxes,

but for slime, as little as 7 per cent. is considered satisfactory in some cases.

The construction of spitzkasten is very simple, and will be readily understood from the example given in figs. 178 and 179.

As it is often impossible to use large spitzkasten consisting of a single pyramid, on account of their great depth and the consequent fall required at the site, the arrangement of a number of small pyramidal boxes, placed side by side and end to end, as indicated in fig. 180, is commonly adopted where large quantities of pulp have to be dealt with.

## CHAPTER XLII.

### COST OF CYANIDE PLANTS.

In speaking of the construction cost of plants, it is customary to take the monthly output in tons as a basis of comparison.

In Johannesburg a single-treatment plant with timber vats and staging on stone foundations, including equipment, costs about £1·25 per ton for, say, 15,000 tons and upwards, and £2·5 for, say, 5000 tons.

In Eastern Australia similar works of about 2000 tons monthly output cost from £2 to £2·3 per ton, according to their situation.

In America a large modern plant, with steel vats fully equipped for classifying and treating battery pulp, can be erected for £2 per ton.

In Johannesburg a modern double-tier steel plant, including tailings wheel and spitzkasten, mechanical haulage, etc., may be priced at £3 per ton treated monthly for 10,000 tons, and £4 for 5000 tons.

In Western Australia a complete outfit for dry crushing and direct treatment inclusive, from rock breaker to filter presses, runs to about £20 per ton of monthly capacity.

Generally this method of stating costs is convenient for the comparison of different plants in a rough-and-ready way, but sometimes it is misleading. For instance, to compare the prices given above for timber and steel plants in Johannesburg, without qualification, would lead to the conclusion that steel construction costs twice as much as timber in that particular district. But there are differences in arrangement and function that have to be taken into account. In the modern plant we have first of all provision for double treatment, which somewhat increases the vat capacity from, say, 0·33 ton for each ton treated per month in the older type, to 0·46 ton in the modern plant. Also, in the latter the double-tier arrangement not only means a greatly increased cost for the supports of the upper vats, but usually involves the addition of a tailings wheel. Therefore, although in comparing the two systems as a whole, it is quite fair to quote prices of construction per ton of monthly treatment, yet in making comparisons of the several methods of construction, it is necessary to analyse the total cost and to compare the several items under equal conditions.

We shall therefore deal with the following matters separately, and give rules and figures for ascertaining as nearly as possible the cost of each, and the limits of its variation.



Excavations.

Foundations and supports.

Vats.

Precipitation plant.

Machinery.

Buildings.

Tailings wheels and pumps.

When formulæ are given for estimating costs, the same letters will be used for the same quantities as in Chapter XXXIV., as for instance C = cubic feet. In order that the figures for labour may be applicable in any country, the coefficients for wages are worked out at one penny per hour, so that W in the formulæ will represent wages expressed in pence per hour. Also L will always mean the cost in pounds sterling.

We may note here, once for all, that the rules and prices given can only be approximate, and are only intended for use when detailed quantities and current local quotations are not available.

**Excavation.**—The cost of this item seldom reaches to more than 4 per cent. of the total, and it therefore needs only brief consideration. Upon gently sloping or flat ground, under average conditions of soil, the following figures will apply to plants of the type shown in figs. 30 to 37, 41 and 42.

Excavation in cubic yards per ton treated monthly :—

Plants without collecting vats, . . .	0·5	cubic yard
Plants with collecting vats, . . .	0·75	„
Two-tier double-treatment plants, . . .	0·75	„
Double-treatment, not superposed, . . .	1·00	„

On flat sites with rock at surface the cost is a minimum, because practically no excavation is required. As a rule, the steeper the ground, the greater is the amount of excavation necessary.

Table LXXVI. gives average prices for this work in several countries, with cost of unskilled labour and supervision added to make it of wider application.

TABLE LXXVI.—*Cost of Excavation in Pence per Cubic Yard, and Labour in Pence per Hour.*

	Soft Ground.	Hard Ground.	Rock with Blasting.	Unskilled Labour per hour.	Super- intendence per hour.
Africa, . . . .	15	20	42	4	18
Australia, . . . .	12	17	36	8	9
America, . . . .	15	25	56	14	17
Siberia, . . . .	5	7	18	1·5	6
Local prices, . . . .					

**Timber Supports for Vats.**—As these are only used for small plants, we need only consider the cost of them in cases where 4 or 5 leaching vats of 16 to 24 feet are used, with corresponding solution tanks. Then the quantity of timber in the supports all taken together per ton of monthly capacity is—

	Cub. ft.	Sq. ft. 1 in. thick
For 16-ft. leaching vats, . . .	0·063	0·75
„ 20- „ „ . . .	0·083	1·00
„ 24- „ „ . . .	0·104	1·25

The prices of timber are given in Table LXXXI., and the cost of framing and erection may be put at

$$L = CW \times 0\cdot0025 = SW \times 0\cdot00021$$

**Masonry Foundations.**—In Table LVIII. some information has been given as to quantities of masonry in several ordinary types of construction, which will be of use in estimating costs, but it is possible to generalise still further for the purpose of minimising labour, in the preparation of preliminary estimates. With the simplest form we may say that in a single-treatment sand plant, for leaching vats only, built on stone foundations, as in figs. 30 to 33, the quantity will vary from 0·1 to 0·2 cubic yard per ton treated monthly. That is to say, for a 4000-ton plant, the masonry under the leaching vats will be somewhere between 400 and 800 cubic yards, say 600. For single treatment of battery pulp, three or four collecting vats are usually provided, therefore their foundations will contain from 0·05 to 0·13 cubic yard per ton. For the double-tier plants, as in figs. 41 and 42, the amount will be from 0·15 to 0·3, and for double treatment, where the vats are not superposed, it will be from 0·2 to 0·4. Then, for shed foundations, solution vats, sumps and sundries, we may add from 0·05 to 1 cubic yard per ton of monthly capacity.

The mean values of these figures, which are derived from actual practice, are collected in Table LXXVII., by the use of which, with Table LXXVIII., a

TABLE LXXVII.—*Summary of Masonry Quantities in Cyanide Plants.*

Description of Plant.	Leaching Vats.	Collecting Vats.	Solution Vats, Shed, and Sundries.	Total.
Single-treatment plant for accumulated tailings, . . . . .	0·15	...	0·08	0·23
Direct treatment of battery pulp with separate collecting vats, . . . . .	0·15	0·09	0·08	0·32
Double treatment with superposed vats, . . . . .	0·23	...	0·15	0·38
Double-treatment plant, vats not superposed, . . . . .	0·15	0·15	0·15	0·45

*Note.*—The monthly capacity multiplied by the figures in the last column gives the approximate total masonry in cubic yards.

fair estimate of the cost of stone foundations, under average conditions, can be made.

TABLE LXXVIII.—*Cost of Hammer-dressed Rubble Masonry, with Materials and Labour for same, and Price of Bricks.*

	Good Bricks per thousand.	Lime per ton of 2240 lbs.	Sand per cubic yard.	Portland Cement per barrel.	Stone per yard measured in place.	Mason's labour per hour.	Coursed rubble per cubic yard, including cement pointing.
Johannesburg, . . .	100/	£2 15 0	4/6	52/	7/	30d.	30/
Victoria and N.S.W., .	32/	2 6 0	2/	15/	4/6	15d.	22/6
U.S.A., . . . . .	30/	2 10 0	...	11/	...	30d.	...
Ural District, Siberia, .	31/	12 0 0	1/	22/	2/	3d.	20/
Local prices, . . . .							

In reference to this last table, it is worth noting that although the individual items vary considerably, yet the cost of the finished work only varies from 20s. to 30s. per yard.

**Iron and Steel Supports.**—In cases where the sizes of vats have been decided upon, as well as the average span of the joists and girders, the weights of the several parts of the framing can be quickly found from Tables LVI., LIX., and LX. Then these weights, multiplied by the respective prices per ton, will give a very close approximation to the total cost of the framing. The following examples will illustrate the use of the tables above referred to, the weights being here worked out in tons of 2240 lbs.

(a) A vat 40 feet in diameter and 10 feet deep weighs 15·6 tons ; allowing 9 inches for filter and for empty space at the top, its contents at 120 lbs. per cubic foot are 622·7 tons, or altogether 638·3 tons. To carry this at a height of 14 feet with 14-foot spans for joists and girders, will require the following amount of steel and iron, according to Tables LIX. and LX. :—

Joists, 14·04 tons  
Girders, 7·95 „ Total steel, say 22 tons.  
Columns—cast iron, 3·90 tons.

(b) When a lower vat is carried in the same way, the lower columns, which may be taken at 6 feet in height, have to carry the two vats and their contents, two sets of joists and girders, and the upper columns, so that the total load is 1324·5 tons, and we have—

Joists and girders as before, 22 tons  
Columns, 2·65 tons

(c) For a vat 20 ft. in diam.  $\times$  6 ft. deep, which weighs 2.85 tons and holds 88.35 tons, the weights for 12-foot spans and columns 10 feet high are—

Joists for upper vat,	.	.	.	1.61 tons
Girders „ „	.	.	.	0.87 „
Columns „ „	.	.	.	0.34 „

(d) For a similar lower vat on 6-foot columns we have—

Joists and girders, as before,	2.48 tons
Columns (carrying 187.7 tons),	0.88 „

As a means of still further saving labour in rough estimates, these weights may now be expressed as percentages on the quantity of sand contained in the vats; reckoned in short tons, as is usual for treatment purposes, the 40 ft.  $\times$  10 ft. vat holds 520 tons, and the 20 ft.  $\times$  6 ft., say, 74 tons.

Therefore, for single-tier plants, the weight of joists only varies from (a) 2.70 per cent. to (c) 2.18 per cent.

For double-tier plants the figures are—

Joists, as before	
Girders,	1.53 per cent. to 1.18 per cent.
Columns (a + b),	0.63 „ to (c + d) 0.49 „

By selecting a suitable figure between these limits, in each case the quantities can be at once deduced from the actual holding capacity of the plant for the material to be treated.

The prices for girders and columns vary so considerably from time to time that it is always advisable, even for rough estimates, to obtain a current quotation if possible.

For cases, however, where this is not possible, we suggest the following average prices, based on figures kindly supplied by Messrs P. & W. Maclellan, of Glasgow.

Prices f.o.b. English ports.

Plain rolled steel joists up to 12 in. $\times$ 6 in., section cut to length and bevelled,	£7 15 0 per ton
Compound riveted girders, with rolled joists and top and bottom plates, each under two tons in weight,	9 15 0 „
Cast iron columns up to 20 ft. long,	6 15 0 „

Then ocean freight, import duties, landing and forwarding charges, inland carriage, and any other costs peculiar to each country must be added.

The erection of such framing may be taken in most cases at 30 per cent., reckoned on the cost of the material delivered at the site.

The following example shows the total cost in Johannesburg for plain steel joints :—

	per ton
Price f.o.b. as above,	£7 15 0 of 2240 lbs.
Freight and insurance to Durban,	1 15 0 „
Railway carriage, 5/9 per 100 lbs.,	6 9 0 „
Receiving, forwarding, etc.,	0 4 0 „
	<hr/>
	£16 8 0

**Cost of Timber Vats.**—The quantity of material can be found by the formulæ given in Chapter XXXIV., or by taking it out in detail from working drawings. The following costs of dressing timber and erecting are based on data in Laxton's *Price Book*, checked and supplemented by the results of actual work in various countries.

For pricing the labour, the most convenient form in which to work out the quantities is generally in running feet of 9 in.  $\times$  3 in. stuff.

Then the cost in pounds, when the whole work is done by hand, is—

$$L = \frac{FW \times 0.3}{240}$$

When the material is machine-dressed, the cost depends upon other matters in addition to the price of labour, but it is sufficiently accurate to assume the total cost as being proportional to wages in different countries. Then price of machine-dressing is—

$$L = \frac{FW \times 0.05}{240}$$

When first-class work is desired, the cost of finishing by hand must be added, viz.—

$$L = \frac{FW \times 0.09}{240}$$

The total cost, therefore, for timber, machine-dressed and hand-finished on the abutting edges, is less than half that when hand labour alone is available, and is—

$$L = \frac{FW \times 0.14}{240}$$

All the above prices are for timber dressed on both faces as well as the edges.

In estimating the cost of erection the same method may be followed, but the running feet of timber in the joists must be included. Then cost of erection is—

$$L = \frac{FW \times 0.072}{240}$$

This includes fixing of hoops and discharge doors. The cost of ironwork material can be obtained by multiplying the weights calculated, as in Chapter XXXIV., page 255, by the prices given in Table LXXIX., which also contains costs of discharge doors, distributors, and filter cloths.

In this and the following tables we have left a blank line for local prices, where current quotations at any particular place can be entered, preferably in pencil.

The following prices (page 352) of complete tanks, f.o.b. San Francisco, are taken by permission from the catalogue of the Pacific Tank Coy., to whom we are also indebted for various other quotations included in Table LXXIX.

TABLE LXXIX.—Average Prices of Materials used in Construction and Equipment of Vats.

Distributors for 30 ft. vat, each.	£ 2 16 25 18 ...
Distributors for 20 ft. vat, each.	£ 3 13 20 15 ...
Hessian or Burlap, in pence per square yard.	4 8 6 ...
Coir Matting, in pence per square yard.	28 30 48 ...
Painting, in pence per square yard.	12 18 16 ...
Discharge Doors, each.	£ s. 3 0 5 5 3 0 4 5
Cast Connections, in pence per lb.	2·50 4·50 2·50 4·00
Round Iron Hoops, bent and screwed, in pence per lb.	2·50 3·50 2·25 3·75
Oregon Pine, in pence per sq. ft. 1 inch thick.	1·68 3·56 1 ...
Oregon Pine, in pence per foot cube.	20·16 42·7 12 ...
Oregon Pine 9 in. × 3 in., in pence per foot run.	3·78 8 2·3 ...
Baltic Deals, in pence per square foot 1 inch thick.	2·89 3·56 ... 0·71
Baltic Deals, in pence per foot cube.	34·7 42·7 ... 8·5
Baltic Deals 9 in. × 3 in., in pence per foot run.	6·5 8 ... 1·6
	Victoria and N.S.W., . Johannesburg, . U.S.A. Ports, . Ekaterinburg, . Local Prices, .

TABLE LXXX.—*Cost of Timber Solution Tanks in America.*

Capacity in gallons.	Diameter in feet.	Height in feet.	Thickness in inches.	Estimated Weight in lbs.	Price in pounds sterling.	Price in dollars.
450	5	4	2	460	3·75	18·00
650	6	4	2	560	5·00	24·00
1,550	8	5	2	970	6·67	32·00
4,000	10	8	2	1,816	11·46	55·00
10,000	18	12	2	3,517	22·92	110·00
20,000	18	12	3	8,400	50·00	240·00
30,000	19	16	3	11,523	67·71	325·00
40,000	21·8	16	3	14,125	83·33	400·00
50,000	24	16	3	15,881	88·54	425·00

The above tanks are made with slight taper and hoops driven on, lugs being provided only on the top hoop.

TABLE LXXXI.—*Cost of Timber Leaching Vats in America.*

Capacity in tons.	Outside Diameter in feet.	Outside Height in feet.	Thickness in inches	Approximate Weight in lbs.	Price in pounds sterling.	Price in dollars.
5	8	4	2	920	6·25	30·00
10	9	5	2	1,260	9·58	46·00
15	11	5	2	1,650	11·46	55·00
20	13	5	2	2,050	14·58	70·00
25	14·5	5	3	3,350	22·92	110·00
40	18	5	3	4,750	31·25	150·00
50	20	5	3	5,640	37·50	180·00
60	22	5	3	6,495	44·79	215·00
100	26	6	3	9,480	62·50	300·00
200	36	6	3	15,800	108·33	520·00
500	48	8	3	28,400	208·33	1000·00
500	52	7	3	30,500	208·33	1000·00

The above tanks are built with vertical sides and lugs on all the hoops. The tons represent 25 cubic feet. Discharge doors and filter bottoms not included.

In the last two tables the figures represent maximum prices, because changes in the lumber market are provided for by a variable discount.

**Masonry Vats.**—These are so seldom built at the present time that it is not necessary to discuss their cost at length. It can, however, be found from the prices of materials given in Table LXXVIII.

**Cost of Steel Vats.**—As a rule, it is cheaper to import steel tanks than to construct them at the place where the plant is to be built. In countries

where protective tariffs are in force, the cost of local manufacture is generally kept close up to that of the imported article, and therefore the cost of steel tanks for most mining districts may be found in the following manner.

1. Find cost f.o.b. in country of origin.
2. Add freight to port nearest to works.
3. Import duties.
4. Inland transport and charges.
5. Erection.

The first item may be taken at £12, 10s. per ton of 2240 lbs. as an average price.

The second, third, and fourth must be ascertained for each mining district.

As an example, we may work out the cost of a vat, say, 30 ft. × 8 ft., manufactured in England and delivered in Johannesburg.

Cost f.o.b. English port, as above, . . .	£12	10	0	per ton
Freight, etc. to Delagoa Bay, . . .	2	0	0	„
Railage to Johannesburg, . . .	5	12	0	„
Sundry charges, . . .	0	8	0	„
	£20	10	0	„

The cost of erection depends principally upon the price of labour, and is given for several countries in Table LXXXII.

TABLE LXXXII.—*Cost of erecting Steel Tanks.*

	Wages in shillings per day of 8 hours.			Cost of Erection, pounds per ton.
	Riveter.	Heater and Holder-up.	Labourers.	
Johannesburg, . . .	20	15	3	9
Victoria and N.S.W., . .	12	8	7·5	7·75
West Australia, . . .	15	12·5	10	10·5
United States, . . .	12·5	11·25	11·25	10·5
„ „ . . .	\$3	\$2·5	\$2·5	\$50
Local Prices, . . .				

**Precipitation Apparatus.**—Rules for finding the quantities of timber and iron in wooden boxes have been given in Chapter XXXVI. The prices for materials in Table LXXIX. will be applicable in this case also, but the cost of preparation and construction is relatively higher for boxes than for vats.



As this is not a large item in the total cost of plants, it is not necessary to analyse it, and the following formula will suffice for the total labour :

$$L = CW \times 0.0127$$

As an example, we may take the electrical box shown in figs. 112 to 118, whose gross capacity is 38 ft.  $\times$  6 ft. 1 in.  $\times$  2 ft. 11 in. = 674.64 cubic feet. Then, by rules on p. 284,  $674.64 \times 0.35 = 236$  cubic feet of timber. Also  $674.64 \times 0.91 = 614$  lbs. of iron. The total cost of the box is then made up of three items, which, at Australian prices current when the box was built, will be—

Material, 236 c.f. Oregon pine at 16.5 pence, . . . . .	£16 4 6
614 lbs. round iron, screwed, and nuts at 2.5 pence, . . . . .	6 7 11
Labour, $236 \times 15d. \times 0.0127$ , . . . . .	44 19 2
	<hr/>
	£67 11 7

This method of estimating is, of course, suitable for either zinc or electrical boxes. For the former the cost is then complete, but for the latter a set of electrodes is often included, although these should properly be charged in the working expenses. The copper connecting-strips, mercury, and other necessities for carrying the current should, for convenience, be included in the electrical apparatus, and not in the cost of the box.

For sheet steel single-compartment boxes, previously described, the average price per cubic foot of zinc space is \$8.50 in America, or, say, £1, 15s.

**Electrical Apparatus for Precipitation.**—This cost is fairly proportional to the anode surface, and the anode surface depends upon the degree of exhaustion required.

The price of electrical apparatus, including dynamo, meters, cables, regulators, and box connections (exclusive of mercury) is—

At Johannesburg prices, 8d. for each square foot of effective anode surface ; and at Australian prices, 6d. per square foot.

For fixing same, add 1d. and  $\frac{1}{2}$ d. per square foot respectively.

**Engines, Boilers, Tram lines, etc.**—These items of cost depend upon so many circumstances, apart from the size of the works, that we have not attempted to generalise upon them. The prices collected in Table LXXXIII., together with what has been said in earlier chapters in connection with these matters, will be useful for estimating purposes.

The labour and material required for unloading, assembling, and erecting engines and boilers, including steam connections, will generally be well covered by an allowance of 20 to 25 per cent. on the cost of these machines.

**Pumps and Piping.**—These may be assumed to bear some relation to the quantity of solution handled per day, though, of course, not a strictly proportional one. Still, as the items are not large, a rough approximation is

TABLE LXXXIII.—*Approximate Prices for Engines, Boilers, and Tramway Material suitable for Cyanide Plants.*

16 ft. Side Tipping Trucks, each.	£	s.	d.	£	s.	d.	£	s.	d.
				14	0	0	11	0	0
20 ft. Side Tipping Trucks, each.	£	s.	d.	£	s.	d.	£	s.	d.
				16	0	0	12	10	0
Portable Turntables, 18 in. gauge, each.	£	s.	d.	£	s.	d.	£	s.	d.
				8	0	0	8	10	0
Points and Crossings, 2-way, 14 lb. rails, each.	£	s.	d.	£	s.	d.	£	s.	d.
				7	0	0	8	0	0
Points and Crossings, 2-way, 12 lb. rails, each.	£	s.	d.	£	s.	d.	£	s.	d.
				6	0	0	7	0	0
14 lb. Track, with sleepers, fish plates, and bolts, £ per mile.				395			500		
12 lb. Track, with sleepers, fish plates, and bolts, £ per mile.				365			465		
Steel Sleepers for 18 in. gauge, each.	£	s.	d.	£	s.	d.	£	s.	d.
				2	0		1	6	
14 lb. Rails, in £ per ton.	£	s.	d.	£	s.	d.	£	s.	d.
				9	10	0	13	0	0
12 lb. Rails, in £ per ton.	£	s.	d.	£	s.	d.	£	s.	d.
				10	0	0	14	0	0
Boilers, 12 to 30 N.H.P., in £ per N.H.P.				12.5			20		
Engines, 12 to 30 N.H.P., in £ per N.H.P.				12			20		
	Victoria and N.S.W.,								
	Johanneburg, .								
	U.S.A. Ports, . .								
	English Ports, .								
	Local Prices, . .								

sufficient, and the following figures, deduced from actual plants, are suggested for the purpose :—

For pumps—

Tons of solution per day  $\times 0.25$  = cost in £ in Australia  
 „ „ „  $\times 0.30$  = „ „ Johannesburg

For piping—

Tons of solution per day  $\times 0.75$  = cost in £ in Australia  
 „ „ „  $\times 1.00$  = „ „ Johannesburg

**Cost of Sheds.**—The rules given in Chapter XXXVII. and the prices in Table LXXXIV. will suffice to estimate the cost of sheds of G. C. iron on light timber framing. For heavy framing, such as is used in America and some other countries, the timber and bolts may be taken at twice the quantities found by the rules.

The labour on framing will be—

$$L = CW \times 0.003$$

and in fixing G. C. iron, including cost of screws and washers, it will be about threepence per square of 100 feet, for each penny in the workman's hourly wage. The approximate total price so found includes ordinary doors and windows.

The material and labour on trestle staging are about the same prices as the framing of sheds. The quantities may be found in the manner suggested in Chapter XXXVIII.

TABLE LXXXIV.—*Prices of Materials used in Construction of Sheds.*

	Building Timber, pence per foot cube.	Building Timber, pence per square foot 1 inch thick.	G. C. Iron, No. 24 Gauge, £ per ton.	G. C. Iron, No. 26 Gauge, £ per ton.	Doors 6 ft. 8 in. $\times$ 2 ft. 8 in. $\times$ 1½ in., each.	Doors 7 ft. $\times$ 3 ft. $\times$ 1½ in., each.	Windows 6 ft. 2 in. $\times$ 3 ft. 6 in. outside dimensions, each.	Cement rendered Concrete Floor 6 inch thick, per square yard.
Victoria and N.S.	14	1.2	£ s. d. 19 0 0	£ s. d. 19 2 6	14	20	29	6/8
Wales,								
Johannesburg, .	42.7	3.56	26 0 0	26 5 0	15	21	36	14/
U.S.A., . . .	12.00	1.00	...	...	...	...	...	...
Ekaterinburg, .	7.2	0.60	22 0 0	22 0 0	...	...	...	...
Local Prices, . .								

**Cost of Tailings Wheels and Pumps.**—The following figures represent the average prices in Johannesburg for timber wheels suitable for lifting pulp

from 100 stamps, or say 5000 tons of pulp daily. The prices include foundations, launder framing, and erection, and were obtained by plotting a number of actual prices, drawing a mean curve, and scaling off from the curve. They therefore show the price at which such wheels ought to be built in that district, but to show the possibility of variation it may be mentioned that in one case the cost was twice the tabulated figure.

20 feet diameter, . .	£725
25    "           . .	900
30    "           . .	1085
35    "           . .	1291
40    "           . .	1536
45    "           . .	1846
50    "           . .	2256
55    "           . .	2811
60    "           . .	3567

The corresponding cost for a tailings pump for the same quantity of pulp is about £1400, so it follows that for lifts less than 40 feet a wheel can be installed more cheaply, while for very high lifts the pump offers a much greater economy in first cost.

**Design, Supervision, etc.**—The cost of drawings, setting out works, and engineering supervision is usually between 3 and 5 per cent. of the total.

When only a rough estimate has been made, as for instance by the rules and prices given in this chapter, a sum of 10 per cent. should be added for contingencies and sundries. When, however, the estimate has been worked out in detail from complete drawings and local current quotations, an allowance of 5 per cent. is sufficient for this purpose.

In some cases where the works are built at a distance from the office of the company, which often happens, in Australia for instance, an allowance of  $\frac{1}{2}$  to 1 per cent. may be made for postages, telegrams, bank charges on transfer of money, travelling expenses, and the like.

The following applications of the rules given in the preceding pages to the case of actual plants will serve to illustrate their use.

*Example I.*—An Australian plant built about 100 miles from Sydney, designed to treat daily 60 tons, measured at 29 feet per ton, with seven days' treatment. Taking 26 working days per month, and one vat discharged per working day, we may state the capacity of the plant, for purposes of comparison, at 1885 tons, of 24 cubic feet, per month. It consisted of six leaching vats 18 ft. in diameter with 8 ft. 6 in. staves, three collecting vats of the same size, one solution vat 18 ft.  $\times$  5 ft., two ditto 15 ft.  $\times$  5 ft., and three intermediate vats 15 ft.  $\times$  5 ft. The contents of the leaching and collecting vats together = 16,600 cubic feet. The plant is of the type shown in figs. 181 to 187.

*Excavations:*

1885 tons monthly  $\times$  .75 yd. = 1414 yds., half being in rock.

707 yds. at 3/ . . . . .	£106	1	0	
707    "    1/5, . . . . .	50	1	7	
	<hr/>			
		£156	2	7
	<hr/>			
Carry forward,	£156	2	7	

Brought forward, £156 2 7

*Masonry:*1885 tons  $\times$  .32 yd. = 603.2 yds. at 22/6, . . . . . 678 12 0*Leaching Vats:*By formula, p. 254, running ft. of 9 in.  $\times$  3 in. = 1047.

Joists, 176 ft.; filter, 26 ft.; total, 1249 ft. at 3.1 pence

per foot . . . . . £16 2 8

Machining and dressing staves, 1047', . . . . . 9 8 2

Iron hoops, 1500 lbs. at 2½d., . . . . . 15 12 6

Cast iron, 1000 lbs. at 3d., . . . . . 12 10 0

Discharge door, . . . . . 3 0 0

Erection, . . . . . 5 12 5

Filter cloths, . . . . . 3 9 8

Painting, 54 yds. at 1/, . . . . . 2 14 0

Six leaching vats at £68, 4s, . . . . . £68 4 0 409 4 0

*Collecting Vats:*

Staves and bottom, 1243'; joists and filter, 202'; total,

1445 ft. at 3.1 pence, . . . . . £18 18 4

Machining and dressing staves and bottom, 1243', . . . . . 10 17 6

Hoops, discharge door, etc., as before, . . . . . 31 2 6

Distributor, . . . . . 18 0 0

Erection, . . . . . 6 10 1

Filter cloths, . . . . . 3 9 8

Painting, . . . . . 3 9 0

Three collecting vats at . . . . . £37 1 8 261 5 0

*Solution and Intermediate Vats:*

Calculated in same way, . . . . . 196 10 0

*Precipitation Boxes:*

Each box contained 281 cub. ft.

Timber 281  $\times$  .35 = 98.35 cub. ft. at 16½d., . . . . . £6 15 8Labour on same, 98.35  $\times$  15  $\times$  .0127, . . . . . 18 14 8

Bolts, etc., 281 lbs. at 2½d., . . . . . 2 18 6

Painting, 20 yds. at 1/, . . . . . 1 0 0

Four precipitation boxes at . . . . . £29 8 5 117 18 8

*Electrical Apparatus for 5040 sq. ft. of anode:*Dynamo, meters, conductors, etc., 5040  $\times$  6½d., . . . . . £136 10 0

Electrodes, sacking, and fixing, 5040 at .048, . . . . . 216 14 5

353 4 5

*Engine and Boiler:*

In this case extra power was provided with a view of subsequent extension of works.

Engine, 8 N.H.P. at £14, . . . . . £112 0 0

Boiler, 12 ,, at £12, 10s., . . . . . 150 0 0

Fixing same, at 22½ per cent., . . . . . 60 0 0

322 0 0

Carry forward, £2494 11 8

Brought forward, £2494 11 8

*Shed*, at 80 tons solution per day :

From Table LXIV. floor space required is 2050 sq. ft. Height, 12 ft. at sides and 18 feet at ridge. Shed should therefore contain 30,750 cub. ft. In the case in point the shed measured 88 ft. × 25 ft., and contained 33,000 cub. ft.

By rules on p. 297 and prices in Table LXXXIV. the cost will be :

297 cub. ft. timber at 13d., . . . . .	£16 1 9	
363 lbs. bolts, etc., at 2½d., . . . . .	3 7 4	
7260 lbs. = 3·24 tons G. C. iron at £16, . . . . .	51 16 10	
Labour on framing, 297 × 15 × ·003, . . . . .	13 7 4	
Labour on covering, 3·24 × 15 × 46d., . . . . .	9 3 4	
Seven doors and twelve windows, at, say, £1 each fixed, . . . . .	19 0 0	
Painting and sundries, say, . . . . .	10 0 0	
		122 16 7

*Staging over Treatment Vats :*

283 ft. run × 3 = 849 cub. ft. at 13d., . . . . .	£45 19 9	
Labour, 849 × 15 × ·003, . . . . .	38 4 1	
Bolts and sundries, say, . . . . .	10 0 0	
		94 3 10

*Pumps and Piping :*

Pumps for 80 tons per day, . . . . .	£20 0 0	
Piping " " . . . . .	60 0 0	
Fixing same, at 25 per cent., . . . . .	16 0 0	
		96 0 0

*Tram Line :*

600 yds. of 18 in. track, at £375 per mile, . . . . .	£127 16 10	
Two sets points and crossings, at £6, 10s., . . . . .	13 0 0	
Fixing same, at 20 per cent., . . . . .	28 3 3	
		169 0 1

*Trucks :*

Ten side-tipping wagons, at £15, . . . . .	150 0 0	
		£3126 12 2
Freight, transport, and insurance—actual payments, . . . . .	583 1 0	
		£3709 13 2
Contingencies, 10 per cent. on £3710, . . . . .	371 0 0	
		£4080 13 2
Office disbursements, 1 per cent. on £4080, . . . . .	40 16 0	
Drawings and engineer's supervision, 4 per cent. on £4080, . . . . .	163 4 0	
		£4284 13 2

Ledger cost of plant, less material returned unused, £4310, 17s. 1d., so that the difference between estimate and actual cost is less than 1 per cent.

## CHAPTER XLIII.

### COST OF TREATMENT.

SINCE the introduction of the cyanide process there has been a continual reduction in the working costs, due principally to the decreasing price of potassium cyanide and to improvements in the methods of handling the material.

The following Witwatersrand costs, which are exclusive of depreciation and general charges, will serve to show how great this reduction has been for one particular district.

1893, average working costs,	.	.	.	4s. 8d.
1895,       "       "	.	.	.	3s. 3d.
1897,       "       "	.	.	.	3s.
1899,       "       "	.	.	.	2s. 6d.

Since the resumption of work after the war the conditions have been abnormal, and it remains to be seen whether the last figure admits of any further reduction.

One of the chief factors, of course, which affect the cost at any particular plant is the scale upon which the operations are conducted.

We shall first separately discuss the principal items which make up the total cost, and then give some examples of the actual detailed figures at various works.

It would greatly facilitate the study and comparison of working expenses if they were all analysed on the same lines, but in the reports published by various mining companies in different countries there are great diversities as regards the grouping of the separate items.

In order to simplify comparison of results, we shall as far as possible arrange all the examples under the following heads and in the same order, with subdivisions when required.

1. Cyanide.
2. Other chemicals, such as lime, caustic soda, etc.
3. Material for precipitation. Zinc, lead, and iron.
4. Power. Coal or firewood, and engine-drivers.
5. Salaries and wages, *i.e.* works manager and shiftmen.
6. Filling and discharging vats, etc.
7. Stores.
8. Maintenance and repairs.

9. Assaying and realising bullion.
10. Lighting.
11. Sundries.

The above items constitute the actual working cost.

12. Depreciation.
13. General charges.
14. Royalties.

The addition of these last three figures make up the total cost of treatment.

The main advantages of keeping and publishing detailed accounts of cost are—

1. To ascertain that no bogus payments are made.
2. To call attention to those items which are largest, with a view to possible reduction.
3. To exhibit any increase or decrease of cost, in any department of the work, in order that the cause of either may be investigated. This often leads to more economical working.
4. To compare results with other plants working under the same or different conditions, and consequently—
5. To ascertain what degree of technical excellence in the operations will give the greatest nett profit.

For small works it is not necessary to go so much into detail as it is for large ones. The larger the plant and the more complicated the processes employed, the more necessary it becomes to minutely subdivide the working costs, not necessarily for publication in monthly statements or annual reports, but for the use of the works manager and consulting metallurgist, and hence for the benefit finally of the shareholders. We know by experience how difficult it sometimes is to persuade accountants, auditors, and directors that cost analyses of this kind are likely to increase dividends. On the Witwatersrand fields at the present day, however, this is fully recognised.

**Cyanide.**—We have already in many places referred to the variations and limitations of the quantity consumed, and the principal use of this item in the monthly accounts is to show whether the actual consumption has been greater than the proper amount, as determined by testing or by previous work. The price of cyanide has decreased almost continually since the introduction of the process, and may now be taken at one shilling per pound, practically in any part of the world. The reason for this uniformity of price is, that the transport charges are very small relatively to the original manufacturer's price. Of course, import duty must be added, where this is charged.

Referring, then, to the minimum consumption for clean tailings, we see that at the present time the lowest cost to be expected under any circumstances is 3d. per ton. But we do not know of any case of practical treatment where this minimum has been reached. A normal figure for what is generally considered clean sand is about 6d. per ton with 0·1 per cent. solution, or 8d. with 0·25 solution.



**Other Chemicals.**—For treatment purposes fat lime has, in many localities, almost displaced caustic soda for neutralising acidity and preparing the ore for treatment. Its cost does not usually differ much from that of building lime, which is given in Table LXXVIII. On tailings, crushed from a quartz reef in granite, and which had undergone very perfect vanner concentration before cyanide treatment, the cost with lime at £2 per ton was 0·3 pence per ton of 24 cubic feet of sand. In another case, with partially decomposed ore containing copper, treated without previous concentration, the cost of lime at £4 per ton was 14·79 pence. These may be taken as the extreme limits of cost for lime, because in cases of extreme acidity it would probably be cheaper to use some caustic soda. The prices for the latter are given in Table LXXXV.

**Cost of Zinc Precipitation.**—This item consists almost entirely of the price of the zinc and the labour for cutting it into shavings. The attendance on the boxes is generally performed by the man who looks after the solutions, and occupies only a small portion of his time. A white labourer can turn about 70 lbs. and a Kafir about 40 lbs. of shavings per day by hand, but except where labour is unusually cheap, it is customary nowadays to use a self-acting lathe, such as described on page 129, so that the turner can also attend to other matters. The prices of sheet zinc and shavings are given in Table LXXXV.

Assuming an average consumption of 0·28 lb. at a mean price for shavings of 6d. per lb., we arrive at about the real cost of zinc precipitation, namely, 1·68 pence per ton of *solution* under normal conditions. In analyses of cost this item generally appears at less than one penny, because the zinc is charged at the purchase price, and no allowance is made for labour in turning, as this is charged in with other labour. In precipitation by the zinc-lead couple there is also the slight cost of lead acetate to be considered, and what is of more importance, the labour of preparing the zinc. Probably 2d. per ton of solution is a fair average figure for the cost of precipitation by zinc-lead couple. This figure is often considerably exceeded in attempting to bring solutions down to 1 or 2 grains per ton.

**Electrical Precipitation.**—With the Siemens-Halske process there are three separate items of cost, namely—

Consumption of the iron anodes, and coverings of same.

Loss of part of the cost of lead foil.

Fuel for supplying current.

The consumption of lead and iron has already been given, and the fuel consumed has been found to be from 12 to 18 lbs. of Transvaal coal or 0·008 to 0·010 cord of dry firewood per ton of solution.

The loss in sacking, connecting strips, sewing and bracing together only amounts to 0·66 penny per lb. of iron, so it is not necessary to consider any slight variations in different countries for this part of the cost.

TABLE LXXXV.—*Prices of Materials used in Precipitation and Refining.*

Litharge, pure, pence per lb.	6-50	7-50	4-5	\$0-09
Fused Borax, per 112 lbs.	65/	...	126/	\$30-24
Borax, per 112 lbs.	28/	34/	32/6	\$7-84
Soda Bicarbonate, per 112 lbs.	10/	19/6	...	...
Caustic Soda, in 5 cwt. drums, pence per lb.	1-50	3-00	1-80	\$0-07
Sulphuric acid, pence per lb.	...	3-75	2-00	\$0-04
Coke, per bag.	3/	8/6	2/8	\$0-64
Sheet Iron, pence per lb.	1-18	2-00	1-08	\$0-02
Lead Foil, pence per square foot.	0-74	0-96	0-6	\$0-01
Lead Foil, pence per lb.	3-86	5-00	3-00	\$0-06
Zinc Shavings, pence per lb.	6-50	5-00	7-50	\$0-15
Sheet Zinc in casks, pence per lb.	2-79	3-81	3-38	\$0-08
Plumbago Crucibles, pence per No.	...	5-50	5-00	\$0-10
Bone Ash, per 112 lbs.	45/	...	23/	\$5-60
	Victoria and N.S.W.,	.	.	.
	Johannesburg,	.	.	.
	U.S.A. Ports,	.	.	.
	" "	3,	.	.
	Local Prices,	.	.	.

Summarising first at Johannesburg prices, we have—

Lead foil, 0·3 lb. at 5d., . . . . .	1·50 pence	
less 25 per cent. of value recovered, . . . . .	·38 „	
	<hr/>	1·12 pence
Iron, 0·5 lb. at 2d., . . . . .	1·00 „	
Covering and attachments, 0·5 lb. at 0·66d., . . . . .	0·33 „	
Fuel, 15 lbs. coal at 17/ per short ton, . . . . .	1·53 „	
	<hr/>	
Total, per ton of solution, . . . . .	3·98 „	

At Australian prices with wood fuel :—

Lead foil, 0·3 lb. at 3·86d., . . . . .	1·16 pence	
less 50 per cent. recovered, . . . . .	·58 „	
	<hr/>	0·58 pence
Iron, 0·5 lb. at 1·18, . . . . .	0·59 „	
Covering and attachments, as above, . . . . .	0·33 „	
Fuel, 0·009 cord at 15/, . . . . .	1·62 „	
	<hr/>	
Total, per ton of solution, . . . . .	3·12 „	

**Cost of Charcoal Precipitation.**—From data published by John I. Lowles\* and W. B. Gray† we have calculated the cost of the various items, in terms that may be readily compared with similar figures given for other methods of precipitation.

Labour in shifting tubs, cleaning-up, and burning off charcoal, 2d. per ton of solution.

Cost of charcoal per oz. of gold, 2·75d., or say for 3½ dwt. solution 0·5 pence per ton.

Therefore the total cost per ton of solution, apart from smelting and refining, is about 2½d.

**Fuel for Pumping, etc.**—In giving the following figures as to fuel consumption it is necessary to mention, that the coal referred to has just about half the heat value of good Welsh steam coal, so that a ton of it is about equal to a cord of average firewood.

The consumption is given in decimals of a long ton, so that it can be also used for a cord of wood.

In a single-treatment plant the amount of coal consumed for pumping solutions is 0·0045 ton per ton of ore treated.

In a double-treatment plant for pumping only, 0·0054 ton.

In a slimes plant for agitation and pumping, 0·0179 ton.

In a single-treatment sand plant, for pumping solutions and hauling material up to vats built on a level site, 0·0060 ton.

For lifting battery pulp to a height of 60 feet by a tailings pump, the consumption of coal per ton of ore crushed is 0·00201 ton.

Each of the above figures, multiplied by the price of a ton of coal or a cord of wood, will give cost for their respective work.

\* *Trans. Inst. Min. and Met.*, vol. vii. p. 190 *et seq.*

† *Trans. Australian Inst. Min. Eng.*, vol. v. p. 141.

**Works Management and Supervision of Treatment.**—The staff for this purpose includes a manager, who is, or should be, a metallurgical chemist or an engineer with some knowledge of metallurgical chemistry, and three shiftmen, each shiftman having charge of the work for eight hours, and being responsible for the proper performance of all operations connected with the treatment, apart from the filling and emptying of the vats. Where white labour only is available, one labourer is also required on day shift for plants over 5000 tons per month, and two men for more than 10,000 tons per month, for cutting zinc, filling boxes, carrying material, etc. With Kafira, the extra labour on actual treatment operations is about one boy for each 5000 tons treated monthly, and one extra shiftman for plants treating more than 10,000 tons.

The ordinary prices paid for this work in several countries is given in Table LXXXVI., and the average cost per ton, calculated according to the number of men above stated, is given in Table LXXXVII.

These figures are, of course, applicable to either sand or slime plants. Where both are worked together under one manager, but with separate shiftmen, the cost for each will be somewhat reduced.

TABLE LXXXVI.—*Wages of Managers and Shiftmen, in £ per month.*

Locality.	Manager.	Shiftman.	Labourer.
Victoria and N.S.W., . .	£15 to £20	£10 to £12	£8
West Australia, . . .	£25 to £40	£22, 10s.	£15
Johannesburg, . . . .	£40 to £60	£15 to £25	£4, 10s.
U.S.A., . . . . .	£20 to £40	£16	£16

TABLE LXXXVII.—*Cost of Labour for Actual Treatment.*

Locality.	Monthly Capacity of Plant in tons.								
	1,000	1,500	2,000	2,500	5,000	7,500	10,000	15,000	20,000
	pence	pence	pence	pence	pence	pence	pence	pence	pence
Australia (Eastern), .	12·12	8·08	6·06	4·85	2·42	1·87	1·40	...	...
West Australia, . .	24·00	16·00	12·00	9·60	5·18	3·46	2·78	1·86	...
Witwatersrand, . . .	...	...	...	10·99	5·50	3·66	3·34	2·30	1·78
U.S.A., . . . . .	18·72	12·48	9·36	7·49	3·74	3·01	2·26	1·76	1·32

**Cost of handling Material.**—The following data will be used in calculating and tabulating some of the costs which fall under this heading.

*Shovelling sand into trucks.*—According to Trautwine, an average man can shovel one cubic yard of sand into a truck in 15 minutes. Assuming as a fair average in cyanide work that the men are only actually shovelling for two-fifths of the working time, this gives 1·8 tons of 24 cubic feet per man per hour, or 0·56 hour per ton.

*Hauling on circular track,* say 200 yards long, by manual labour, including tipping. With white labour, from actual practice, one man can haul and tip 6·04 tons per hour, or at the rate of 0·166 hour per ton. For each additional 100 yards of track add 0·034 hour.

*Shovelling from vats* through bottom discharge doors. Allow 8 tons per man per hour, which is equal to 0·125 hour per ton. This is for vats 8 feet deep. For shallower vats the quality shovelled is reduced roughly in proportion to the depth of the vat.

*Sluicing-out of vats.*—With, say, 5 feet head of water, one man can sluice out about 16 tons of sand per hour = 0·063 hour per ton. With 60-feet head, a 2-inch hose and  $\frac{1}{2}$ -inch nozzle, according to W. Magenau,\* one man can shift 50 tons in  $1\frac{1}{2}$  hours = 0·03 hour per ton.

All the above figures refer to white labour only. With Kafirs, experience shows that each boy averages about two-thirds of the work of a white man.

The following costs have been calculated from the above data.

TABLE LXXXVIII.—*Handling Material.—Labour Costs, in pence per ton.*

Wages, in pence per hour.	Shovelling into Trucks.	Shovelling from Vats through Bottom Discharge.	Sluicing from Vats at Low Pressure.	Sluicing from Vats at High Pressure.	Tramming on Circular Track 200 yards long and tipping.	Add for each extra 100 yards of track.	Tipping on Dump and keeping same in order.
2	1·11	0·25	0·13	0·06	0·33	0·09	0·03
4	2·22	0·50	0·25	0·12	0·66	0·18	0·05
6	3·33	0·75	0·38	0·18	0·99	0·27	0·08
7	3·89	0·88	0·44	0·21	1·16	0·32	0·09
8	4·44	1·00	0·50	0·24	1·32	0·36	0·10
9	5·00	1·13	0·56	0·27	1·49	0·41	0·11
10	5·56	1·25	0·63	0·30	1·66	0·45	0·13
12	6·67	1·50	0·75	0·36	1·99	0·55	0·15
14	7·78	1·75	0·88	0·42	2·32	0·64	0·18
16	8·89	2·00	1·00	0·48	2·65	0·73	0·20

*Note.*—As the figures in this table are for labour only, an extra 10 per cent. may be added for contractor's profit, or for supervision and incidentals when the work is done at day wages. For Kafirs, use the figure in the first column, which is equal to  $1\frac{1}{2}$  times the hourly wage.

\* *Mines and Minerals*, vol. xxi. p. 300.

As an example of the use of the above table, the case of an American plant quoted by W. Magenau,\* where the vats were discharged by shovelling into cars underneath by contract at 10 cents = 5d. per ton, with labour at \$2.50 per day of 8 hours, or 15.6d. per hour. From the lowest line of the table we find that at 16d. per hour the costs are—

Shovelling into cars, . . . . .	2d.	per ton
Tramming, . . . . .	2.65	„
Dumping, . . . . .	0.20	„
Total, . . . . .	4.85	„

Reducing this in the ratio of 16 to 15.6 in order to be exact, we have as cost for labour only, . . .	4.73	„
Add 10 per cent. for contractor's profit, . . .	.47	„
	5.20	„

*Cost of handling material with Filter Presses.*—As the question of filter pressing slimes is being generally discussed at the present time, the following table, giving the labour costs for filling and discharging presses, will be of interest.

The figures are based on actual work done in Western Australia, where three skilled men are employed per day for supervision, and in addition three labourers for each 1000 tons treated monthly. This number of men, of course, includes three shifts of 8 hours each.

TABLE LXXXIX.—*Cost of Labour for filling and emptying Filter Presses, exclusive of tramming cakes to dump.*

	Skilled Labour, per 8 hour shift.	Unskilled Labour, per 8 hour shift.	Tons of Dry Slime treated per month.				
			2000	3000	4000	6000	8000
Western Australia, . . .	15/	11/8	20.69	18.00	16.65	15.30	14.62
Eastern Australia, . . .	8/	7/	11.88	10.44	9.72	9.00	8.64
Witwatersrand, . . . .	18/	3/	12.78	9.72	8.10	6.48	5.67
America, . . . . .	12/6	10/6	18.09	15.84	14.72	13.59	13.03

*Note.*—By putting Kafir labour on the Rand at 3s. per 8 hours, sufficient allowance is made for the inferior quality of native labour, the actual cost in food and wages being about 3s. for 12 hours.

*Mule Haulage.*—The following figures are deduced from work on a large scale with 20-foot trucks on 18-inch track.

\* *Eng. and Min. Jour.*, vol. lxx. p. 70.

When speed is desired, it is better to have a separate animal and driver for each truck, as an average speed of 4 miles an hour can then be maintained, allowing for ordinary delays, and also 5 minutes for filling and 2 minutes for tipping. This is the usual method of working on short lines where a large number of trucks would be troublesome. With longer lines more trucks can be used, so that speed of travelling is of less importance, and the work can then be done more cheaply by giving two or three trucks to each mule and driver, and hauling at, say, 3 and  $2\frac{1}{2}$  miles per hour respectively. All these speeds and loads are for an approximately level road, and the results are tabulated below.

TABLE XC.—*Cost of Animal Haulage in pence per ton of 24 cubic feet, including Driver. The distances include the return track for empty trucks.*

Cost of Mule and Driver in pence per hour.	Hauling Single Trucks.			Two Trucks at a time.			Three Trucks at a time.		
	500 yards.	1000 yards.	1500 yards.	1000 yards.	1500 yards.	2000 yards.	1500 yards.	2000 yards.	2500 yards.
8	1·80	2·48	3·18	2·19	2·48	2·94	2·21	2·57	2·94
10	2·25	3·10	3·96	2·74	3·10	3·67	2·76	3·22	3·67
12	2·70	3·73	4·75	3·28	3·72	4·41	3·32	3·86	4·41
14	3·15	4·35	5·54	3·83	4·35	5·14	3·87	4·51	5·14
16	3·60	4·97	6·33	4·38	4·97	5·89	4·42	5·15	5·88
18	4·05	5·59	7·12	4·92	5·59	6·61	4·97	5·79	6·61

We may now estimate, by way of example, the cost of handling material in a double-treatment plant with superposed vats at Johannesburg prices, with a run of, say, 1000 yards on the residue track. Kafirs at 3s. 4d. per day of 10 hours, including food and quarters, = 4d. per hour  $\times$  1·5 = 6d., therefore for manual labour the third line of Table LXXXVIII. is to be used. The cost of mule and driver is taken at 14d. per hour in this case, therefore we have—

Transferring from top vat,	. . .	0·75 pence.
Shovelling from lower vat,	. . .	0·75 „
Tramming to dump,	. . .	4·35 „
Dumping,	. . .	0·08 „
		<hr/> 5·93 „
Add 10 per cent.,	. . .	·59 „
Total,	. . .	<hr/> 6·52 „

*Rope Haulage.*—In this system of transport there is necessarily a certain minimum number of men employed, irrespective of the quantity of material or the distance run, say—

- 1 Engine-driver.
- 1 Fireman.
- 1 Labourer at each end of line.
- 1 Overseer, when native labour is employed.

On this basis the figures in Table XCI. have been worked out for those countries in which at present this method is much used. When power can be obtained without a special engine for the haulage a considerable economy is effected, as shown.

TABLE XCI.—*Labour Costs for Rope Haulage, in pence per ton.*

	Tons hauled per day.						
	400	500	600	700	800	900	1000
Johannesburg, Special Engine, . .	1·32	1·06	0·88	0·75	0·66	0·59	0·53
„ Borrowed Power, . .	0·69	0·55	0·46	0·39	0·35	0·31	0·28
U.S.A., Special Engine, . . . .	1·00	0·80	0·67	0·56	0·50	0·44	0·40
„ Borrowed Power, . . . .	0·63	0·50	0·42	0·36	0·31	0·28	0·25

With ordinary colonial coal at 20s. per ton, the fuel costs may be taken at 0·08 to 0·11 pence per 100 yards of line.

Repairs to machinery of this class amount to about 6 per cent. per annum on its first cost, and renewal of ropes from 50 to 100 per cent. per annum.

**Cost of Roasting.**—With reverberatory furnaces worked by hand rabbling, the cost of white labour is just about equal to one man's daily wage per ton of ore.

With mechanical furnaces this is reduced by about 40 per cent., but with native labour the saving is much less, because the cost of skilled supervision is then the chief item.

The cost of fuel may be roughly estimated by assuming a consumption of  $\frac{1}{3}$  ton of coal or  $\frac{1}{3}$  cord of wood for each ton of ore roasted.

These, and the maintenance of furnaces, are the chief items when roasting for subsequent cyaniding.

In America, for hand reverberatories, the cost for fuel and labour is from 15s. 6d. to 17s. 6d. per ton, and in Johannesburg from 7s. to 8s.

**Stores.**—The materials included under this heading vary considerably, according to the manner in which the analysis of costs is made. Thus, oil and waste may be included in this item or in the cost of power. Materials for lighting are sometimes included in general stores, and sometimes given separately. Again, all chemicals except cyanide are often charged to stores account, and consequently we find in the Witwatersrand district alone, such remarkable variations, as from 0·255 to 3·701 pence per ton, in contemporaneous published statements from different plants.



**Maintenance and Repairs.**—The cost of these depends upon the price of labour and materials in the same way as the cost of construction, and may therefore, for estimating purposes, be stated as a percentage of the latter, in which case we find, from the examination of many actual returns, that the figure will generally lie between 0·72 and 0·91 per cent. per annum, calculated on the original cost of the plant.

Insurance may also be fairly charged under this head, although this is not usually done.

**Clean-up and Realisation of Gold.**—Some idea of the cost of this may be gathered from the examples of complete costs given below. But these figures are all calculated per ton of material treated, and it is often necessary to know the cost per ounce of gold recovered; for instance, when comparing different methods of precipitation or of refining.

The following costs for cleaning-up and smelting precipitates, from zinc boxes in America, were given some years ago by W. R. Ingalls,\* but are still valid for the same method, namely, acid treatment and pot smelting at the present time.

		Cents per oz. Pence per oz.	
		Fine Gold.	Fine Gold.
Labour cleaning-up, 15 hours at 30 c.,	\$4·50	... 2·25	... 1·125
Acid, fluxes, and supplies, . . . .	8·00	... 4·00	... 2·000
Coke, 400 lbs. at 7½ c., . . . .	3·00	... 1·50	... 0·750
Total for 200 oza., . . . .	\$15·50	... 7·75	... 3·875

For materials alone, the figure is therefore 5·5 cents = 2·75d. The cost of the smelter's time is not given, and, in fact, in small plants this work is generally done by the manager or by the assayer, and is included, therefore, in other items.

The cost of smelting the gold contained in the ash from charcoal precipitation, according to J. I. Lowles in the paper previously quoted, is 1s. per ounce of bullion 900 fine, or 13·33d. per ounce of fine gold. This figure is for fluxes, fuel, and pots at Australian prices, but does not include labour.

E. H. Johnson † published the following figures for a clean-up similar to Ingalls', in Johannesburg. Dry weight slime, 504 lbs.; after acid treatment, 100 lbs.

		Per oz. Fine Gold.
672 lbs. acid at 4½d., . . . .	£12 12 0	... 4·88 pence
66 lbs. borax at 37/6 per cwt., . .	1 2 11	... 0·44 "
9 lbs. carb. soda at 2½d. per lb., .	0 1 10	... 0·04 "
9 lbs. fluor spar at 4d., . . . .	0 3 0	... 0·06 "
5 bags coke at 8/6, . . . .	2 2 6	... 0·82 "
No. 60 crucible, . . . .	1 7 6	... 0·58 "
Total for 620 oza. fine gold, . .	£17 9 9	... 6·77 "

\* "Precipitation of Gold from Cyanide Solutions," *Min. Ind.*, vol. iv. p. 338.

† *Jour. Chem. and Met. Soc. of S. Africa*, June 19, 1897.

For smelting zinc-gold slimes without acid treatment in Johannesburg, G. T. M. MacBride gives the following :\*

			Per oz. Fine Gold.
Borax, . . . . .	£1 5 0	...	0·52 pence
Soda, . . . . .	0 6 7	...	0·14 „
Fluor spar, . . . . .	0 5 6	...	0·11 „
Nitre, . . . . .	0 2 6	...	0·05 „
Pots, . . . . .	1 1 3	...	0·44 „
Coal, . . . . .	1 16 0	...	0·75 „
Labour, . . . . .	1 2 8	...	0·47 „
Total for 580 oza. fine gold, . . . . .	£5 19 6	...	2·48 „

Say 2d. per oz. for materials, and 2½d. including labour.

The following costs for two methods of refining precipitates from the zinc-lead couple at the present time in Johannesburg have been kindly supplied by P. S. Tavener.

1. Sludge treated by sulphuric acid, followed by calcining and pot smelting :

			Per oz. Fine Gold.
Sulphuric acid, 721 lbs. at 3½d., . . . . .	£11 5 6	...	5·69 pence
Stores, crucibles, clay liners, etc., . . . . .	11 0 0	...	5·56 „
Coal, . . . . .	2 18 6	...	1·48 „
Total for 475 oza. fine smelted gold, . . . . .	£25 4 0	...	12·73 „

2. Materials used for smelting and cupelling the product from zinc-lead precipitation boxes, at Bonanza mine, Johannesburg, from June to September 1902:

			Per oz. Fine Gold.
Coal, . . . . .	£82 18 1	...	1·54 pence
Coke, . . . . .	9 8 6	...	0·18 „
Fireclay, . . . . .	3 0 0	...	0·06 „
Firebricks and slabs, . . . . .	17 15 6	...	0·33 „
Paper bags, . . . . .	1 17 6	...	0·08 „
Lead, . . . . .	3 0 8	...	0·06 „
Crucibles and liners, . . . . .	1 15 0	...	0·03 „
Bone ash, . . . . .	10 0 0	...	0·19 „
Sundries, . . . . .	2 0 4	...	0·04 „
Total for 12,810 oza. fine gold, . . . . .	£131 10 7	...	2·46 „

Any of the above figures can, of course, be readily converted into cost per ton of material treated for any given extraction per ton, either actual or estimated.

The last figures, with the exception of one or two trifling details, will apply to the melting and cupellation of the gold-coated lead foil from the Siemens and Halske process, in the same district.

From small plants the lead bullion can be sent to customs works for cupellation. In Johannesburg the costs for this are from 4d. to 9d. per oz. of fine gold, less £5 per ton for litharge.

\* *Proc. Chem. and Met. Soc. of S. Africa*, vol. i. p. 337.

In Australia, customs works buy the bullion outright at the following prices :

Fine gold, assay value at 82/ per oz.  
 Silver 90 per cent. of assay, at bar silver price  
 Lead—London price soft lead, less 7½ per cent.

**Depreciation.**—The object of this charge, which is, perhaps, more truly expressed by the word ‘amortisation,’ is to replace from revenue the money originally spent in construction of plant. Consequently, it is arrived at by dividing this amount by the total tonnage to be treated during the whole life of the plant. It is possible, of course, that the plant may have some value at the end of its life, but, as a rule, it is better to neglect this in calculating the amortisation charge, except in special cases of temporary plants, which are afterwards to be removed and used again by the same owners. For if a plant has to be sold after several years’ work, the price obtained for the saleable parts is negligible in comparison with the first cost of construction.

A single example will sufficiently illustrate the principle above stated.

Assuming the case of a plant to treat 20,000 tons monthly for ten years, which cost, say, £50,000. This sum, then, divided by the total quantity to be treated, namely, 2,400,000 tons, gives 5d. per ton as the proper charge for depreciation.

In the case of a mine, the total tonnage has to be estimated roughly at first, but the charge may be adjusted from time to time, according to the reserves of ore opened up. In very doubtful cases, it is better to start with a relatively high charge, based on a conservative estimate of the material available for cyanide treatment, and to reduce the charge afterwards when a larger quantity of material is assured.

**General Charges.**—This item has little interest from a metallurgist’s point of view, because it depends upon factors entirely beyond his control. It may include a calculated portion of all or any of the following expenses :

General manager’s salary.

Consulting engineer’s salary.

Mine office expenses.

Head or other office expenses.

Directors’ fees, etc.

**Examples of Working Costs—Australia.**—(a) Average costs of working a small plant with electrical precipitation in New South Wales, treating 46,400 short tons of clean tailings in 2½ years :

1. Cyanide, . . . . .	6·65d.
2. Lime, . . . . .	0·29
3a. Lead foil, . . . . .	0·41
3b. Iron anodes, . . . . .	1·02
4. Firewood, at 15/ per cord, . . . . .	2·10
5. Manager and shiftmen, . . . . .	9·18
6. Filling and emptying vats, . . . . .	7·54

Carry forward, 2a. 8·14d.

	Brought forward, 2s. 3'14d.
7a. General stores, . . . . .	1'50
7b. Electrical stores, . . . . .	0'31
8. Maintenance and repairs, . . . . .	0'41
9. Transport and sampling lead bullion, . . . . .	0'64
11. Sundries, . . . . .	0'82
	<hr/>
	Working cost, . . . 2s. 6'82d.
12. Depreciation, . . . . .	8'32
13. General charges, . . . . .	4'16
	<hr/>
	Total cost, . . . 3s. 7'30d.

(b) This example is from a plant in Victoria, worked under the same management as (a), also with electrical precipitation, but on tailings containing decomposed copper minerals. The figures refer to an average treatment of about 850 tons per month. The price for iron anodes is estimated, but all the other figures represent the actual outlay for each item, and is the average of 7 months' work. A comparison of (a) and (b) shows how very greatly the total cost depends upon the nature and quantity of the material treated.

1. Cyanide, . . . . .	3s. 7'74d.
2. Other chemicals, . . . . .	1s. 2'86
3a. Lead foil, . . . . .	2'03
3b. Iron anodes, . . . . .	1'87
4. Firewood, . . . . .	6'23
5. Manager and shiftmen, . . . . .	2s. 11'67
7. Stores, . . . . .	9'35
8. Maintenance and repairs, . . . . .	1'35
9. Transport and realisation of bullion, . . . . .	5'50
11. Sundries, . . . . .	0'92
	<hr/>
	Working cost, . . . 11s. 9'79d.
12. Depreciation, . . . . .	2s. 0'00
13. General charges, . . . . .	4'90
14. Royalties, . . . . .	7'33
	<hr/>
	Total cost, . . . 14s. 10'02d.

(c) Cost at Myall's United mine in New South Wales\* when working about 2000 long tons per month. We give in the second column the figures worked out per short ton.

	Long ton.	Short ton.
1. Cyanide, . . . . .	1s. 1'7d. ...	1s. 0'23d.
2a. Lime, . . . . .	0'5 ...	0'45
2b. Chemicals, . . . . .	0'5 ...	0'45
3. Zinc, . . . . .	1'7 ...	1'52
4. Steam, . . . . .	1'2 ...	1'07
5. Supervision and boys, . . . . .	4'7 ...	4'20
6a. Filling vats, . . . . .	1s. 0'3 ...	10'98
6b. Extra labourers, . . . . .	1'0 ...	0'89
6c. Tip timber, . . . . .	0'1 ...	0'09
9. Coke, . . . . .	0'6 ...	0'54
11. Sundries, . . . . .	0'5 ...	0'45
	<hr/>	<hr/>
	Working cost, . . . 3s. 0'8d. ...	2s. 8'87d.

\* W. R. Thomas, *Trans. Inst. of Min. and Met.*, vol. vii.

It may be noted that the cost of stores has apparently been distributed and included in other items, and that no charge is made for maintenance and repairs.

The next four examples were quoted by S. J. Truscott from the respective Companies' Annual Reports, at a meeting of the Chemical and Metallurgical Soc. of S. Africa.\*

(d) Cost at Ivanhoe mine, Kalgurli, W.A., for year 1901, for about 4000 tons of *sand* per month by double treatment after wet crushing :

1. Cyanide, . . . . .	1s. 5'69d.
2. Lime, . . . . .	2'44
3a. Zinc, . . . . .	0'70
3b. Cutting zinc, . . . . .	0'29
4a. Fuel, . . . . .	6'46
4b. Engine-driving, . . . . .	2'33
5. Superintendence, . . . . .	2'11
6a. Filling and emptying, . . . . .	1s. 2'94
6b. Removal of old tailings, . . . . .	4'72
7. Stores, . . . . .	1'91
8. Repairs and renewals, . . . . .	4'67
9a. Assays, . . . . .	1'66
9b. Smelting, . . . . .	0'53
11. Condensed water, . . . . .	3'37

Working cost, . . . 5s. 3'82d.

(e) Cost at Ivanhoe mine, year 1901, for treatment of about 5000 tons per month of *slime* by agitation and filter pressing :

1. Cyanide, . . . . .	1s. 6'50d.
2. Lime, . . . . .	2'04
3a. Zinc, . . . . .	0'54
3b. Turning zinc, . . . . .	0'22
4a. Fuel, . . . . .	5'88
4b. Engine-driving, . . . . .	1'98
5, 6. Labour, . . . . .	1s. 4'34
6b. Trucking, . . . . .	7'49
6c. Removal of old alimes, . . . . .	7'98
7a. General stores, . . . . .	1'70
7b. Filter cloth, . . . . .	0'86
8. Repairs and renewals, . . . . .	4'09
9a. Assays, . . . . .	2'30
9b. Smelting, . . . . .	0'60
11a. Condensed water, . . . . .	3'22
11b. Salt water, . . . . .	1'51
11c. Compressed air, . . . . .	3'77

Working cost, . . . 6s. 7'02d.

(f) Cost of treating 3300 tons of *sand* per month at Golden Horseshoe mine, Kalgurli, W.A., during year 1901, by double treatment after wet crushing :

\* Jour., vol iii p. 43.

1. Cyanide, . . . . .	1s. 10'82d.
2. Lime and chloride of lime, . . . . .	0'64
3a. Zinc, . . . . .	0'95
3b. Turning zinc, . . . . .	1'06
4a. Fuel, . . . . .	2'51
4b. Engine driving and firing, . . . . .	1'85
5a. Superintendence, . . . . .	1'15
5b. Solutionists, pumping, etc., . . . . .	5'32
6. Filling and emptying, . . . . .	1s. 2'41
6a. Breaking slime and sands, . . . . .	2'90
6b. Extension of tramways and repairs, . . . . .	1'60
6c. Rehandling sands from dump, . . . . .	1'03
8. Repairs and maintenance, . . . . .	5'68
9a. Assaying and retorting, . . . . .	2'34
9b. Realisation charges, . . . . .	4'62
10. Electric light, . . . . .	1'05
11a. Fresh water, . . . . .	2'16
11b. Salt water, . . . . .	2'24
	<hr/>
	6s. 2'33d.

(g) Cost at Golden Horseshoe mine for year 1901, for treatment of 4000 tons of *slime* monthly by agitation and filter pressing :

1. Cyanide, . . . . .	1s. 11'03d.
3a. Zinc, . . . . .	0'95
3b. Turning zinc, . . . . .	0'62
4a. Fuel, . . . . .	9'22
4b. Engine driving and firing, . . . . .	4'66
5. Superintendence and attending solutions, . . . . .	6'03
6a. Breaking slime and sands, . . . . .	6'77
6b. Discharging presses, . . . . .	1s. 1'12
6c. Dumping residues, . . . . .	8'97
7, 8. General stores and charges. Repairs, . . . . .	1s. 3'73
7b. Filter cloth, . . . . .	4'17
9a. Assaying and retorting, . . . . .	2'91
9b. Realisation charges, . . . . .	3'79
11a. Condensed water, . . . . .	7'49
11b. Salt water, . . . . .	4'64
11c. Compressed air, . . . . .	6'99
	<hr/>
	9s. 11'09d.

(h) This example is given to show the increased cost of work in less accessible parts of Western Australia. The figures are from the Report of the Sons of Gwalia, Ltd., for the year 1900, during which period 13,133 tons of accumulated tailings and 28,267 tons of sand from current battery product were treated. Very few details are given in the report.

1-7. Cyaniding, . . . . .	6s. 8'83d.
8. Repairs, . . . . .	2'88
9. Assaying, . . . . .	1'00
13a. Administration, . . . . .	2'70
13b. General expenses, . . . . .	3'97
	<hr/>
	7s. 7'38d.

(2) This example, which shows the whole cost of working in Kalgurli from shaft head to residue dumps, has been compiled from actual returns supplied to us for the purpose. The costs, as far as the separation of sands from slimes, are, of course, common to both products, and are given separately—per ton.

Labour from brace to mill, . . . . .	7 26d.
Crushing,—	
Millmen, feeders, and greasers, . . . . .	6 99d.
Engine-room labour, . . . . .	5 20
Fuel and water, . . . . .	1s. 4 83
Maintenance and repairs, . . . . .	1s. 4 40
Sundry stores, oil, waste, etc., . . . . .	1 10
	<hr/>
	3s. 10 52
Separation of sands and slimes, . . . . .	2 44
	<hr/>
Total cost of preparation, . . . . .	4s. 8 22d.

*Treatment of Sands.*

1. Cyanide, . . . . .	1s. 2 99d.
3. Zinc, . . . . .	1 50
4a. Fuel, . . . . .	6 68
4b. Labour on machinery, . . . . .	4 46
4c. Water, . . . . .	0 17
5a. Works manager, . . . . .	1 81
5b. Shiftmen (solution hands), . . . . .	3 22
6a. Filling vats, . . . . .	1s. 3 48
6b. Emptying vats, . . . . .	9 50
7. Stores, . . . . .	0 78
8. Maintenance and repairs, . . . . .	1s. 1 54
9. Assaying and realisation, . . . . .	6 28
	<hr/>
Total working cost, . . . . .	6s. 6 41d.

*Treatment of Slimes.*

1. Cyanide, . . . . .	1s. 3 21d.
3. Zinc, . . . . .	1 17
4a. Fuel, . . . . .	6 78
4b. Labour on machinery, . . . . .	4 52
4c. Water, . . . . .	0 17
5a. Works manager, . . . . .	1 81
5b. Shiftmen, . . . . .	3 27
6a. Filling vats, . . . . .	6 06
6b. Filling and discharging presses, including trucking, . . . . .	2s. 5 18
7a. Stores, . . . . .	0 79
7b. Filter cloths, . . . . .	2 21
8. Maintenance and repairs, . . . . .	1s. 1 56
9. Assaying and realisation, . . . . .	6 10
	<hr/>
Total working cost, . . . . .	7s. 6 83d.

As the ratio of slime to sand was about 3 to 1, the cost per ton for the whole of the work may be found thus—

1 ton sand at, . . . . .	6s. 5 <sup>4</sup> / <sub>16</sub> d.
3 tons lime, . . . . .	22s. 8 <sup>4</sup> / <sub>16</sub> d.
<hr/>	
For 4 tons crushed, . . . . .	29s. 1 <sup>9</sup> / <sub>16</sub> d.
Thus, cost of cyaniding per ton crushed, . .	7s. 3 <sup>4</sup> / <sub>16</sub> d.
Add for crushing and separation, as above, .	4s. 8 <sup>2</sup> / <sub>16</sub> d.
<hr/>	
Total surface working cost, . . . . .	11s. 11 <sup>7</sup> / <sub>16</sub> d.

Or practically 12s. per ton, exclusive of depreciation and general charges.

(j) Example of costs of the Diehl process at Hannan's, Brownhill, Western Australia, in the month of July 1901, when 2210 tons of ore were treated. These are R. W. Feldtmann's figures, taken from H. Knutsen's \* paper read before the Institute of Mining and Metallurgy.

#### Labour:

3 Solution hands, . . . . .	£79 16 6
3 " " assistants, . . . . .	58 6 8
2 Filter press foremen, . . . . .	46 16 8
4 " cleaners, . . . . .	75 7 8
4 " truckers, . . . . .	64 8 7
1 Watchman, . . . . .	18 3 1
2 Rouseabouts, . . . . .	43 5 6
Proportion timekeeper, . . . . .	6 0 1
" electrician, . . . . .	1 10 0
" carters, . . . . .	3 0 0
" crushing lime, . . . . .	1 0 0

#### Supplies:

Lime, . . . . .	26 10 0
Zinc shavings, . . . . .	17 3 6
Potassium cyanide, . . . . .	418 9 2
Bromo-cyanide, . . . . .	457 0 0
Sulphuric acid, . . . . .	20 13 6
Fresh water, . . . . .	4 9 10
Bromo-salts, . . . . .	109 3 0
Other supplies, . . . . .	78 9 7
Repairs, . . . . .	64 9 2
Power, . . . . .	231 16 9
Assaying, . . . . .	45 6 8
Administration, . . . . .	16 11 8
<hr/>	
Total for 2210 tons, . . . . .	£1887 17 7
" per ton, . . . . .	17 1 <sup>0</sup> / <sub>2</sub> d.

United States.—(k) Example of American costs at Standard Cons. Mines, Bodie, California, arranged from figures published by T. H. Leggett † for 6515 tons treated in 79 days.

\* *The Min. Jour.*, vol. lxxii, p. 1114, 1902.

† *Inst. of Min. and Met.*, vol. iv, p. 151.



2. Lime, 7 55 „ „ . . . . .	0 28
3. Zinc, 0 29 „ „ . . . . .	1 28
4. Firewood, 0 013 cord per ton, . . . . .	5 28
5, 6. Wages, also filling and emptying, . . . . .	2s. 3 10
6a. Hauling, . . . . .	1s. 3 32
7. Stores, . . . . .	1 16
9. Sulphuric acid, . . . . .	0 25
10. Miscellaneous, . . . . .	0 26
<hr/>	
Working cost, 5s. 8 66d.	

This apparently excessive total is accounted for by the high prices of KCy (2s. 6d. per lb.), labour and fuel.

(i) Cost of treating about 3600 tons of sand monthly at Geyser Marion mine \* in 1899.

We have added the costs per ton in pence.

These figures are the averages on 43,271 tons.

	Cents.	s. d.
1. Cyanide, at 30 cents per lb., . . . . .	15	... 7 5d.
2. Lime, at 40 cents per bushel, . . . . .	0 2	... 0 10
3. Zinc, at 9 cents per lb., . . . . .	2 2	... 1 10
4. Coal, at \$5 00 per ton, . . . . .	5 0	... 2 50
5, 6. Labour, including filling vats, . . . . .	12 9	... 6 45
6b. Discharging tailings, . . . . .	10 0	... 5 00
7, 8. Oil, repairs, and incidentals, . . . . .	2 8	... 1 40
9. Proportion of assayer's salary, . . . . .	1 9	... 0 95
<hr/>		
Working cost, 50 0	...	2s. 1d.

*Witwatersrand.*—In this district it is now customary, in preparing the annual reports of the mining companies, to analyse all the treatment costs on the basis of tonnage milled, which is a satisfactory arrangement for giving all necessary information to shareholders. But the metallurgist requires, more particularly, to know the working costs per ton of material actually treated by any process. We have therefore in the following examples re-calculated each item on the basis of tons treated.

Thus the figures in example *c* were calculated from costs and tonnage given in the Reports of the Simmer & Jack Proprietary Mines, Ltd.† As various changes in quantities and methods occurred during the period covered by these tabulated returns, the following explanation of them is necessary.

The first column gives the cost of the Macarthur Forrest process in the early part of 1897, at the rate of about 4000 tons monthly in an old plant.

The second column refers to the treatment of 8000 tons per month in 1897, in a new plant with Siemens-Halske precipitation, and the third column is for the same method from January 1898 to June 1899.

\* *Eng. and Min. Jour.*, W. Magenan, vol. lxx. p. 70.

our disposal by the London Secretary, Mr J. T. Bedborough.

The fourth column shows the same costs as the third, but calculated on the ton of ore milled.

		M'Arthur, 1897. <i>m</i>	Siemens, 1897. <i>n</i>	Siemens, 1898. <i>o</i>	Per ton milled. <i>o</i>
1.	Cyanide, . . . . .	10·13	5·96	4·64	3·33
2.	Chemicals, . . . . .	2·04	0·31	·07	·05
3a.	Zinc, . . . . .	·86	...	...	...
b.	Lead foil, . . . . .	...	2·39	1·22	·87
c.	Mercury, . . . . .	...	...	·03	·02
4a.	Steam coal, . . . . .	3·88	2·66	·73	·52
b.	Electric power, . . . . .	...	4·98	·80	·58
5a.	Foremen, . . . . .	·87	1·96	·30	·22
b.	Shiftmen, . . . . .	4·45	5·12	1·85	1·32
c.	Native labour, . . . . .	7·51	1·47	·85	·61
d.	Compound expenses, . . . . .	2·26	·53	·60	·36
e.	Engine-drivers, . . . . .	1·84	1·24	·42	·30
6.	Contractors, filling, etc., . . . . .	11·09	9·15	6·79	4·86
7a.	Lubricants, . . . . .	·24	·35	·11	·08
b.	Tools and steel, . . . . .	·23	·10	·02	·01
c.	Iron and sheet steel, . . . . .	·11	·09	·21	·15
d.	Machinery renewals, . . . . .	·03	·31	1·22	·87
e.	Ropes, belts, etc., . . . . .	...	·77	·65	·47
f.	Hardware and sundries, . . . . .	·52	·98	·59	·42
g.	Candles, . . . . .	·02	·01	...	...
8a.	Timber, etc., . . . . .	·59	·52	·49	·35
b.	Smithy coal, . . . . .	·27	·12	·10	·07
c.	Blacksmiths and mechanics, . . . . .	·90	1·58	1·15	·82
d.	Carpenters, . . . . .	·18	0·64	·43	·31
e.	Surface contractors, . . . . .	·04	...	·03	·02
f.	Workshop expenses, . . . . .	·20	·14	·17	·12
g.	Insurance, . . . . .	...	·08	·01	...
9a.	Assay office expenses, . . . . .	1·07	1·58	·58	·42
b.	Refining expenses, . . . . .	1·01	1·20	·94	·67
c.	Zinc, . . . . .	...	·04	...	...
10.	Electric light, . . . . .	...	·83	·33	·23
	<b>Total working cost,</b>	<b>4s. 2·29d.</b>	<b>3s. 9·11d.</b>	<b>2s. 1·23d.</b>	<b>1s. 6·05d.</b>
12.	Depreciation, . . . . .	...	...	...	...
13a.	Manager and consulting engineer, . . . . .	·25	·38	·10	·07
b.	Mine office, . . . . .	·49	·46	·10	·07
c.	Stable expenses, . . . . .	2·39	·19	·08	·06
d.	Rent, . . . . .	13·39	...	...	...
14.	Royalty, . . . . .	...	3·78	2·71	1·94
	<b>Total costs,</b>	<b>5s. 6·81d.</b>	<b>4s. 1·87d.</b>	<b>2s. 4·22d.</b>	<b>1s. 8·19d.</b>

(p) The following figures are extracted from the Annual Reports of the Crown Reef G. M. Co., Ltd., issued from 1897 to 1899.\* Each column of figures represents the average of twelve months, and these costs include the treatment of the sands and the spitzlutte concentrates.

\* For these reports we are indebted to the courtesy of the London Secretary, Mr A. Moir.

Report issued in	1897.	1898.	1899.
Tons of Sand and Concentrates,	145,367	139,076	165,087
All labour in working, filling, and emptying, . . . . .	<i>s. d.</i> 1 2'05	<i>s. d.</i> 11'44	<i>s. d.</i> 8'99
All supplies, including cyanide and zinc, . . . . .	1 7'56	1 4'12	1 3'93
Maintenance, . . . . .	5'45	6'36	4'60
Operating solution and vacuum pumps, . . . . .	0'65	0'60	0'75
Lighting, . . . . .	0'42	0'41	0'47
Mechanical haulage from vats, . . . . .	0'36	0'55	0'86
Total working costs,	3 4'58	2 11'48	2 7'10

**Cost of Treating Current Slimes.**—(q) The following figures are from the reports of the Crown Reef G. M. Co., already referred to. Electrical precipitation.

Report issued in	1897.	1898.	1899.
Tons treated.	21,539	40,955	32,111
All labour in working, . . . . .	<i>s. d.</i> 1 5'48	<i>s. d.</i> 11'63	<i>s. d.</i> 1 0'81
All supplies, including cyanide and lead, . . . . .	2 4'90	1 8'52	1 4'94
Maintenance, . . . . .	8'81	10'26	1 1'50
Operating solution and slimes pumps, . . . . .	4'59	3'98	6'40
Lighting, . . . . .	0'59	0'46	0'80
Total working cost,	5 0'37	3 5'85	4 2'45
Royalty, . . . . .	4'06	2'27	2'15
Total cost,	5 4'43	3 8'12	4 4'60

(r) This example is from the Reports of the Simmer & Jack Proprietary Mines, Ltd., to 30th June 1899 :

Tons slime treated in 18 months, 122,635.  
Total cost, £26,480, 16s. 5d.  
Cost per ton, 4s. 3'82d.

(s) The following costs of the whole treatment of dry crushed pyritic ore at Luipaard's Vlei Estate were published by Franklin White :\*

\* *Trans. Inst. Min. and Met.*, vol. vii. pp. 187, 188.

*Cost of Dry Crushing Pyritic Ore with Rolls.*

White wages, . . . . .	10'995d.
Native ,, . . . . .	3'924
Stores, . . . . .	8'863
Steam power, . . . . .	7'858
Electric ,, . . . . .	0'408
Maintenance, . . . . .	7'322
Lighting, . . . . .	0'485
Rock breaker cost, . . . . .	9'747
Total, . . . . .	4s. 1'800d.

*Cost of Cyaniding Dry Crushed Ore.*

1. Cyanide, 0'40 lb., . . . . .	7'70d.
2. Lime, 2'55 ,, . . . . .	1'15
3. Zinc, 0'31 ,, . . . . .	1'08
4. Power and light, . . . . .	2'30
5, 6. White wages, . . . . .	11'18
,, Native ,, . . . . .	7'20
7. Coal, coke, and sundries, . . . . .	1'00
8. Maintenance, . . . . .	3'45
9. Assay charges, . . . . .	3'60
10. Stables, . . . . .	0'85
Total, . . . . .	3s. 3'51d.

(t) The following figures were published by John R. Williams,\* as examples of the lowest costs obtained up to the present time (1902), in working slimes by the decantation process on the Rand :

	Company No. 1.	Company No. 2.	Company No. 3.
	<i>s. d.</i>	<i>s. d.</i>	<i>s. d.</i>
1. Cyanide, . . . . .	5'18	5'30	5'85
2. Lime, . . . . .	6'08	3'54	5'90
3. Zinc, . . . . .	...	0'92	1'81
4. Power, . . . . .	4'20	2'16	1'16
5, 6. Filling and discharging, and } labour, white and native, }	1 1'14	10'63	1 3'77
7. Stores and material, . . . . .	2'88	3'31	0'25
8. Maintenance, . . . . .	0'10	5'07	0'53
9. Assaying and sampling, . . . . .	2'49	1'70	3'24
11. Water, . . . . .	0'13	0'78	...
	2 10'20	2 9'41	2 10'51

\* *Jour. Chem. and Met. Soc. of S. Africa*, vol. iii. p. 61.

## CHAPTER XLIV.

### COMPLETE CYANIDE PLANTS.

No. 1.—In figs. 181 to 188 a complete cyanide plant is illustrated, which, although built for single treatment, is also suitable for double treatment, and is a fair sample of a well built plant with timber vats. These figures are all reproduced from the working drawings. The first two, viz., 181 and 182, show the relative levels of collecting vats *a a*, leaching vats *b b*, with the staging *c c* for transferring the tailings from the former to the latter. Owing to the small fall available below the battery, the collecting vats are sunk partially below the ground level, while the treatment vats are raised on piers to provide for dumping the residues without any second lifting. Figs. 183 and 184 together constitute a plan of the whole plant, showing the position of the storage vats *e e*, intermediate vats *f f*, and precipitation shed *g*. The latter is larger than necessary, because provision was made for the possible addition of boxes for solution from a slimes plant. Figs. 185 and 186 are transverse sections at the places indicated. Fig. 187 is a detail plan of part of the tipping platform above the vats, and fig. 188 is a detail of one roof truss for the shed.

There are six collecting vats, 18 ft. in diameter, with 8 ft. staves, each having a distributor with six arms. There is a double tunnel under the vats, and two discharge doors in each vat.

There are also six leaching vats, 23 ft. in diameter inside, with 8 ft. staves, each having two 12-inch discharge doors. A double track on the inclined road from under the collecting vats to the top of the leaching vats enables two sets of trucks to be worked, so that one set can be filled while the other set is tipped. The hauling is done by a steam winch placed in line with the centre of the two trucks, 48 feet beyond the left-hand vat in fig. 181. The vats are discharged by manual labour, with 20 ft. side-tipping trucks.

There are four upper storage tanks, for alkaline wash, strong and weak KCy solutions, and clean water, respectively. After passing through the vats and precipitation boxes, the solutions are pumped back to the storage tanks, from the sumps *k k*, by four Worthington duplex steam pumps. Of the four precipitation boxes, one is intended for alkaline washes, one for strong solution, and two for weak solutions. The function of the intermediate vats *f f* is to allow of a regular supply of solution to the electrical precipitation boxes, irrespective of the actual quantity coming from the treatment vats at any



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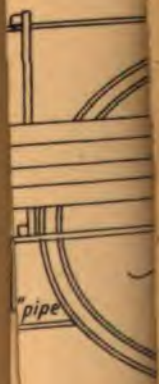
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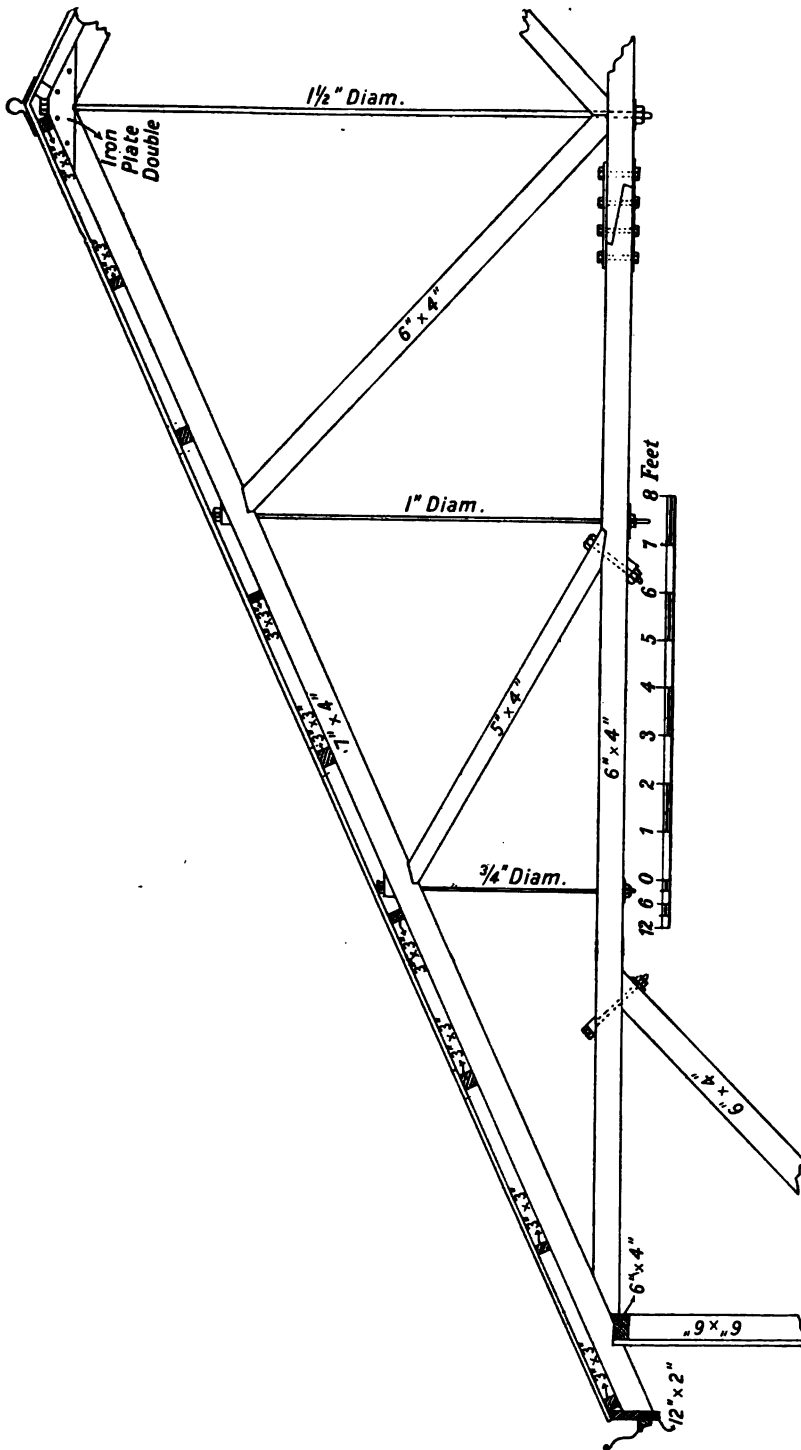
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**FIG. 188.—Detail of Roof Truss.**

moment. To make this supply absolutely regular, the small vats or cisterns *h h* are introduced. Each of these has an ordinary ball cock (made of iron) on the pipe which supplies it from the intermediate vat, so that the level of the solution in the cistern *h* cannot vary more than a couple of inches. Consequently the boxes receive their solution under a practically constant head, and the flow through them is quite uniform.

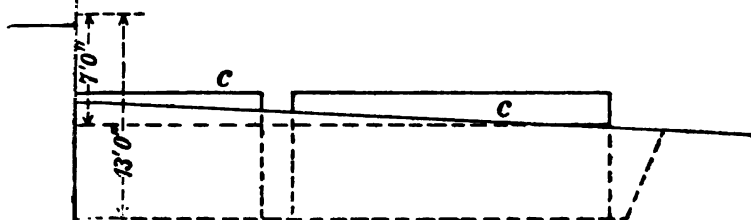
The collecting, leaching, and storage vats are all built of 3-inch Oregon pine, and are placed on timber joists resting on stone foundations. All framing and staging are of Australian hard wood; the trestles have posts 6 in. square, tenoned into 12-in.  $\times$  12-in. sills. At the top the uprights are notched to receive two 9-in.  $\times$  3-in. pieces on edge, one on each side, and these are bolted together with  $\frac{5}{8}$ -in. bolts. There are four 12-in.  $\times$  3-in. bearers, with herring-bone struts at the centre of each span. The staging is decked with 9-in.  $\times$  3-in. deals, with 1-in. spaces between them, except the parts over the vats, which are covered with strips 3 in. deep,  $2\frac{1}{4}$  in. wide at the top, and 2 in. wide at the bottom, laid on edge, with  $2\frac{1}{2}$  in. clear between them. The strips are alternately 6 ft. and 4 ft. long, as shown in fig. 187, and are fastened to the bearers with 6-in. spikes. A handrail and stairway are provided, as shown in the drawings. The plant was designed for 2500 short tons monthly, with six days' treatment.

No. 2.—The plant illustrated in figs. 189, 190, and 191, designed by John Kelly, consulting engineer, Johannesburg, is one of the double-tier type, which has been so largely used on the Witwatersrand. It has very nearly the same vat capacity as the plant previously described, so that the two types may be readily compared. Fig. 189 is a front elevation, with the precipitation shed omitted; fig. 190 is a cross section through vats and sheds; and fig. 191 is a plan.

There are six upper vats *a a*, 24 ft. in diam. by 7 ft. deep, for collection and first treatment of the rough concentrates and sands; these collectors have Butters-Mein distributors and four bottom discharge doors *i i*. The six lower vats *b b*, 24 ft. by 8 ft., wherein the treatment is completed, have each three discharge doors *j j* (two only are shown in the drawing), arranged in one line over a single tunnel *k* between the stone piers *l l*. There are two solution storage vats *c c*, 20 ft. by 8 ft., and one 16 ft. by 8 ft., placed in an excavation below the shed. All the vats are of steel, and the leaching vats are carried on rolled joists, which for the lower vats rest directly on stone piers as at *e e*, while the upper joists *f f* rest upon compound girders *g g*, carried by cast iron columns *h h*.

The battery pulp is elevated by a wheel, whose centre lines are shown on the plan; this wheel delivers through the launder *m* to a spitzlutte *n*, which separates the pyritic material and delivers it to one of the upper vats. The spitzlutte overflow passes by launders to another of the upper vats. From the slat gate discharges of these upper vats the slimy water passes to a spitzkasten *o*, whose overflow goes directly to the slime dam, and whose underflow, containing fine sand, goes back to the tailings wheel, as already described in Chapter XXVI.

[To face page 384.



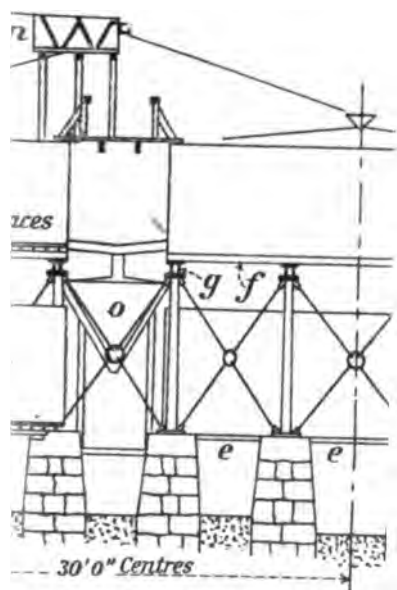
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the 1990s, the number of people in the UK who are aged 65 and over has increased by 1.5 million (1990–1999) and is projected to increase by a further 1.5 million by 2010 (Office of National Statistics 2000). The number of people aged 65 and over who are dependent on others for their care is projected to increase from 1.2 million in 1999 to 2.2 million in 2010 (Office of National Statistics 2000). The number of people aged 65 and over who are dependent on others for their care is projected to increase from 1.2 million in 1999 to 2.2 million in 2010 (Office of National Statistics 2000). The number of people aged 65 and over who are dependent on others for their care is projected to increase from 1.2 million in 1999 to 2.2 million in 2010 (Office of National Statistics 2000).

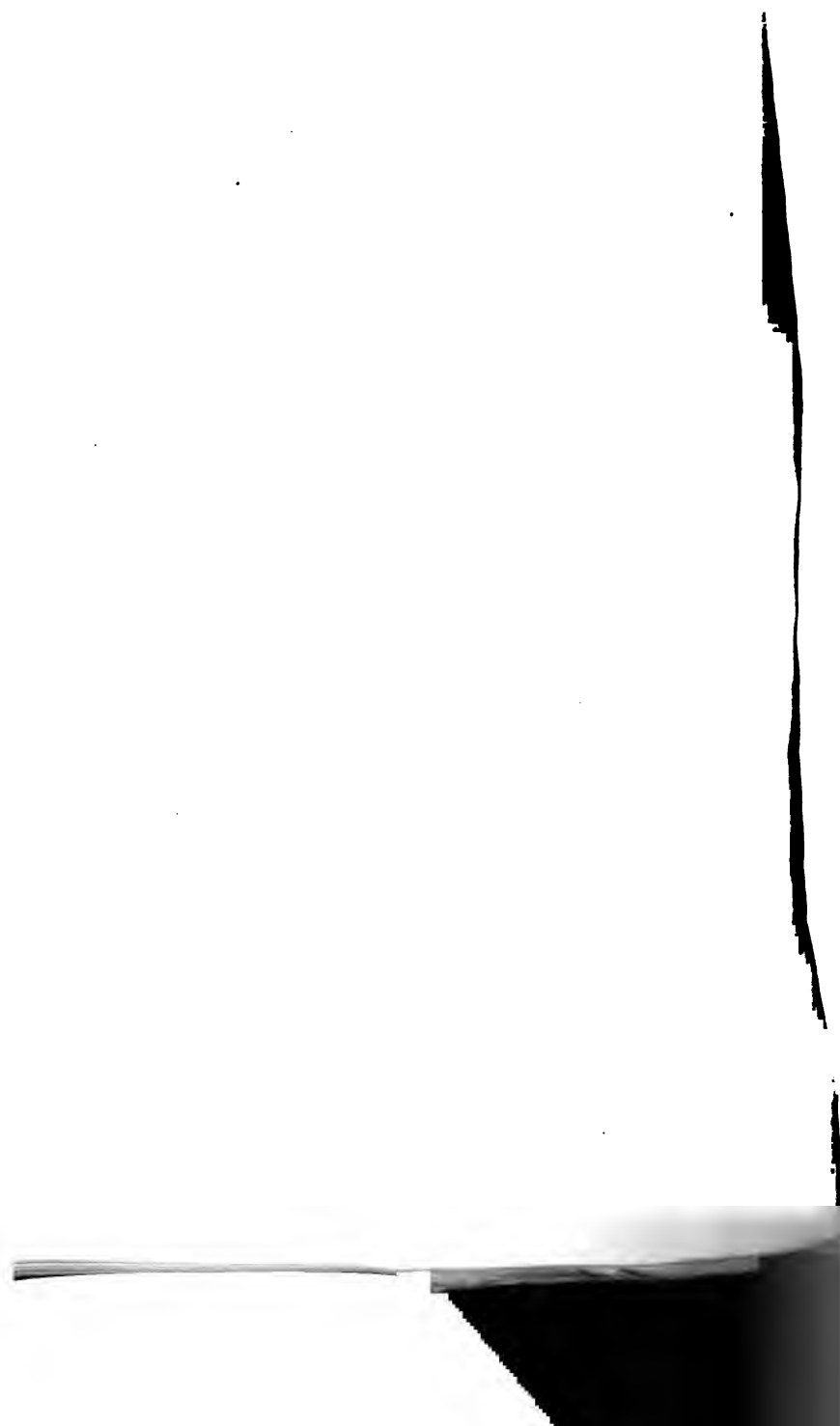
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The solutions pass from the leaching vats to a settling box *p*, which clarifies them to some extent before they enter the precipitation boxes. The half of the settling box through which the solution rises is filled with coir to assist the clarification. From the boxes the solutions gravitate to the storage tanks, and are pumped up again to the leaching vats when required by a 4-inch centrifugal pump *q*.

The tailings wheel is driven by an electrical motor *r* by means of the gearing *ss*. No intermediate vats are provided in this plant.

TABLE XCII.—*Showing Contents of Vats in cubic feet.*

Inside Depth in feet and inches.		Inside Diameter of Vat in feet.														
		4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
4 0	50.3	78.5	118.1	153.9	201.1	254.5	314.2	380.1	452.4	530.9	615.8	706.9	804.2	907.9	1011.1	1114.3
4 3	53.4	88.4	120.2	163.6	213.6	270.4	333.8	403.9	480.7	564.1	654.2	751.0	854.5	964.7	1076.6	1189.1
4 6	56.5	88.4	127.2	173.2	226.2	286.3	353.4	427.7	508.9	597.3	692.7	795.2	904.8	1021.1	1143.1	1270.6
4 9	59.7	98.3	134.3	182.8	238.8	302.2	373.1	451.4	537.2	630.5	731.2	839.4	955.0	1078.1	1207.1	1341.6
5 0	62.8	98.2	141.4	192.4	251.3	318.1	392.7	475.2	565.5	663.7	769.7	883.6	1005.1	1135.1	1272.1	1416.1
5 3	66.0	103.1	148.4	202.0	263.9	331.0	412.3	498.9	593.8	696.8	808.2	927.8	1056.1	1192.1	1335.1	1485.1
5 6	69.1	108.0	155.5	211.7	276.5	349.9	432.0	522.7	622.0	730.0	846.7	971.9	1106.1	1248.1	1397.1	1553.1
5 9	72.3	112.9	162.6	221.3	289.0	365.3	451.6	546.4	650.3	763.2	885.1	1016.1	1156.1	1305.1	1461.1	1624.1
6 0	75.4	117.8	169.6	230.9	301.6	381.7	471.2	570.2	678.6	796.4	923.6	1060.1	1206.1	1362.1	1526.1	1697.1
6 3	78.5	122.7	176.7	240.5	314.2	397.6	490.9	594.0	706.9	829.6	962.1	1104.1	1257.1	1419.1	1589.1	1767.1
6 6	81.7	127.6	183.8	250.2	328.7	413.5	510.5	617.7	735.1	862.8	1001.1	1149.1	1307.1	1475.1	1651.1	1835.1
6 9	84.8	132.5	190.9	259.8	339.3	429.4	530.1	641.5	763.4	895.9	1039.1	1198.1	1357.1	1532.1	1715.1	1906.1
7 0	88.0	137.4	197.9	269.4	351.9	445.3	549.8	665.2	791.7	929.1	1078.1	1237.1	1407.1	1589.1	1776.1	1971.1
7 3	91.1	142.4	205.0	279.0	364.4	461.2	569.4	689.0	820.0	962.3	1116.1	1281.1	1458.1	1646.1	1838.1	2037.1
7 6	94.2	147.3	212.1	288.6	377.0	477.1	589.1	712.8	848.2	995.5	1155.1	1325.1	1508.1	1702.1	1900.1	2104.1
7 9	97.4	152.2	219.1	298.3	389.6	493.0	608.7	736.5	876.5	1029.1	1193.1	1370.1	1558.1	1759.1	1964.1	2175.1
8 0	100.5	157.1	226.2	307.9	402.1	508.9	628.3	760.3	904.8	1062.1	1232.1	1414.1	1608.1	1816.1	2030.1	2250.1
8 3	103.7	162.0	233.3	317.5	414.7	524.8	648.0	784.0	933.1	1096.1	1270.1	1458.1	1659.1	1873.1	2094.1	2322.1
8 6	106.8	166.9	240.3	327.1	427.3	540.7	667.6	807.8	961.3	1128.1	1308.1	1502.1	1709.1	1929.1	2156.1	2390.1
8 9	110.0	171.8	247.4	336.7	439.3	556.7	687.2	831.5	990.0	1161.1	1347.1	1546.1	1759.1	1986.1	2220.1	2462.1

9 0	113.1	176.7	254.5	346.4	452.4	572.6	706.9	855.3	1018	1195	1385	1590	1810	2048	2290
9 3	116.2	181.3	261.5	356.0	465.0	588.5	726.5	879.1	1046	1228	1424	1635	1860	2100	2354
9 6	119.4	186.5	268.6	365.6	477.5	604.4	746.1	902.8	1074	1261	1462	1679	1901	2156	2417
9 9	122.5	191.4	275.7	375.2	490.1	620.3	765.8	926.6	1103	1294	1501	1723	1960	2213	2481
10 0	125.7	196.4	282.7	384.8	502.7	636.2	785.4	950.3	1131	1327	1539	1767	2011	2270	2545
10 3	128.8	201.3	289.8	394.5	515.2	652.1	805.0	974.1	1159	1361	1578	1811	2061	2327	2608
10 6	131.9	206.2	296.9	404.1	527.8	668.0	824.7	997.9	1188	1394	1616	1856	2111	2383	2672
10 9	135.1	211.1	303.9	413.7	540.4	683.9	844.3	1022	1216	1427	1655	1900	2161	2440	2736
11 0	138.2	216.0	311.0	423.3	552.9	699.8	863.9	1045	1244	1460	1693	1944	2212	2497	2799
11 3	141.4	220.9	318.1	433.0	565.5	715.7	883.6	1069	1272	1498	1732	1988	2262	2554	2863
11 6	144.5	225.8	325.2	442.6	578.1	731.6	903.2	1098	1301	1526	1770	2032	2312	2610	2926
11 9	147.7	230.7	332.2	452.2	590.6	747.5	922.8	1117	1329	1560	1809	2076	2362	2667	2990
12 0	150.8	235.6	339.3	461.8	603.2	763.4	942.5	1140	1357	1593	1847	2121	2413	2724	3054
12 3	153.9	240.5	346.4	471.4	615.8	779.3	962.1	1164	1385	1628	1886	2165	2463	2781	3117
12 6	157.1	245.4	353.4	481.1	628.3	795.2	981.8	1188	1414	1659	1924	2209	2518	2837	3181
12 9	160.2	250.3	360.5	490.7	640.9	811.1	1001	1212	1442	1692	1963	2253	2564	2894	3244
13 0	163.4	255.3	367.6	500.3	653.5	827.0	1021	1235	1470	1726	2001	2297	2614	2951	3308
13 6	169.6	265.1	381.7	519.5	678.6	858.8	1060	1283	1527	1792	2078	2386	2714	3064	3435
14 0	175.9	274.9	395.8	538.8	703.7	890.6	1100	1380	1583	1858	2155	2474	2816	3178	3563
14 6	182.2	284.7	410.0	568.0	728.9	922.5	1139	1378	1640	1925	2232	2562	2915	3291	3690
15 0	188.5	294.5	424.1	577.3	754.0	954.3	1178	1496	1696	1991	2309	2651	3016	3405	3817
15 6	194.8	303.3	438.3	596.5	779.1	986.1	1217	1473	1753	2057	2386	2739	3116	3518	3944
16 0	201.1	314.2	452.4	615.8	801.2	1018	1257	1521	1810	2124	2463	2827	3217	3632	4072

TABLE XCII.—continued.

Inside Depth in feet and inches.		Inside Diameter of Vat in feet.																
		19	20	21	22	23	24	25	26	27	28	29	30	31	32			
4 0	1134	1257	1385	1521	1662	1810	1964	2124	2290	2463	2642	2827	3019	3217	3418			
4 3	1205	1336	1472	1616	1766	1923	2086	2256	2433	2617	2807	3004	3208	3418	3619			
4 6	1276	1414	1559	1711	1870	2036	2209	2389	2577	2771	2972	3181	3396	3619	3820			
4 9	1347	1492	1645	1806	1974	2149	2332	2522	2720	2925	3137	3358	3585	3820	4021			
5 0	1418	1571	1732	1901	2077	2262	2454	2655	2863	3079	3303	3534	3774	4021	4222			
5 3	1489	1649	1818	1996	2181	2375	2577	2787	3006	3233	3468	3711	3968	4222	4423			
5 6	1559	1728	1905	2091	2285	2488	2700	2920	3149	3387	3633	3888	4151	4423	4694			
5 9	1630	1806	1992	2186	2389	2601	2823	3063	3292	3541	3798	4064	4340	4624	4925			
6 0	1701	1885	2078	2281	2493	2714	2945	3186	3435	3695	3963	4241	4529	4825	5027			
6 3	1772	1964	2165	2376	2597	2827	3068	3318	3578	3848	4128	4418	4717	5027	5228			
6 6	1843	2042	2251	2471	2701	2941	3191	3451	3722	4002	4293	4595	4906	5228	5429			
6 9	1914	2121	2338	2566	2804	3054	3313	3584	3865	4156	4459	4771	5085	5429	5690			
7 0	1985	2199	2425	2661	2908	3167	3436	3717	4008	4310	4624	4948	5283	5630	5891			
7 3	2056	2278	2511	2756	3012	3280	3559	3849	4151	4464	4789	5125	5472	5831	6092			
7 6	2126	2356	2598	2851	3116	3398	3682	3982	4294	4618	4954	5301	5661	6032	6293			
7 9	2197	2435	2684	2946	3220	3506	3804	4115	4437	4772	5119	5478	5849	6233	6494			
8 0	2268	2513	2771	3041	3324	3619	3927	4247	4580	4926	5284	5655	6038	6434	6685			
8 3	2339	2592	2857	3136	3428	3732	4050	4380	4724	5080	5449	5832	6227	6635	6886			
8 6	2410	2670	2944	3231	3532	3845	4172	4513	4867	5234	5614	6008	6416	6836	7087			
8 9	2481	2749	3031	3326	3635	3958	4295	4646	5010	5388	5780	6185	6604	7037	7390			

9 0	2552	2827	3117	3421	3739	4072	4418	4778	5153	5542	5945	6362	6793	7288
9 3	2623	2906	3204	3516	3843	4185	4541	4911	5296	5696	6110	6538	6982	7439
9 6	2694	2985	3290	3611	3947	4298	4663	5044	5439	5850	6275	6715	7170	7640
9 9	2764	3063	3377	3706	4051	4411	4786	5177	5582	6004	6440	6892	7359	7841
10 0	2835	3142	3464	3801	4155	4524	4909	5309	5726	6158	6605	7069	7548	8042
10 3	2906	3220	3550	3896	4259	4637	5031	5442	5869	6311	6770	7245	7736	8244
10 6	2977	3299	3637	3991	4362	4750	5164	5575	6012	6465	6935	7422	7925	8445
10 9	3048	3377	3723	4086	4466	4863	5277	5707	6155	6619	7101	7599	8114	8646
11 0	3119	3456	3810	4181	4570	4976	5400	5840	6298	6773	7266	7775	8302	8847
11 3	3190	3534	3897	4276	4674	5089	5522	5973	6441	6927	7431	7952	8491	9048
11 6	3261	3613	3983	4372	4778	5202	5645	6106	6584	7081	7596	8129	8680	9249
11 9	3331	3691	4070	4467	4882	5316	5768	6238	6728	7235	7761	8306	8869	9450
12 0	3402	3770	4156	4562	4986	5429	5891	6371	6871	7389	7926	8482	9057	9651
12 3	3473	3848	4243	4657	5090	5542	6013	6504	7014	7543	8091	8659	9246	9852
12 6	3544	3927	4330	4752	5193	5655	6136	6637	7157	7697	8257	8836	9435	10053
12 9	3615	4006	4416	4847	5297	5768	6259	6769	7300	7861	8422	9012	9623	10254
13 0	3686	4084	4503	4942	5401	5881	6381	6902	7443	8005	8587	9189	9812	10455
13 6	3828	4241	4676	5132	5609	6107	6627	7168	7730	8313	8917	9543	10189	10857
14 0	3969	4398	4849	5322	5817	6333	6872	7433	8016	8621	9247	9896	10567	11259
14 6	4111	4555	5022	5512	6024	6560	7118	7698	8302	8928	9578	10249	10944	11662
15 0	4253	4712	5195	5702	6232	6786	7363	7964	8588	9236	9908	10603	11322	12064
15 6	4395	4869	5369	5892	6440	7012	7609	8229	8875	9544	10238	10956	11699	12466
16 0	4536	5027	5542	6082	6648	7238	7854	8496	9161	9852	10568	11310	12076	12868



TABLE XCII.—continued.

Inside Depth in feet and inches.	Inside Diameter of Vat in feet.															
	33	34	35	36	37	38	39	40	41	42	43	44	45	50		
4 0	3421	3632	3848	4071	4301	4536	4775	5027	5281	5542	5809	6082	6362	7854		
4 3	3635	3859	4089	4326	4570	4820	5077	5341	5611	5888	6172	6462	6759	8345		
4 6	3849	4086	4330	4580	4838	5104	5376	5655	5941	6234	6535	6842	7157	8886		
4 9	4063	4313	4570	4835	5107	5387	5674	5969	6271	6581	6898	7223	7555	9327		
5 0	4277	4540	4811	5089	5376	5671	5973	6283	6601	6927	7261	7603	7952	9818		
5 3	4490	4767	5051	5344	5645	5954	6272	6597	6931	7274	7624	7983	8350	10308		
5 6	4704	4994	5292	5598	5914	6238	6570	6912	7261	7620	7987	8363	8747	10799		
5 9	4918	5221	5532	5853	6182	6521	6869	7226	7591	7966	8350	8743	9145	11290		
6 0	5132	5448	5773	6107	6451	6805	7168	7540	7922	8313	8713	9123	9543	11781		
6 3	5346	5675	6013	6362	6720	7088	7466	7854	8252	8659	9076	9503	9940	12272		
6 6	5559	5901	6254	6616	6989	7372	7765	8168	8582	9005	9439	9883	10388	12763		
6 9	5773	6128	6494	6871	7258	7655	8063	8482	8912	9352	9802	10264	10785	13254		
7 0	5987	6355	6735	7125	7526	7939	8362	8796	9242	9698	10165	10644	11133	13745		
7 3	6201	6582	6975	7380	7795	8222	8661	9111	9572	10044	10528	11024	11531	14235		
7 6	6415	6809	7216	7634	8064	8506	8959	9425	9902	10391	10892	11404	11928	14726		
7 9	6629	7036	7456	7888	8333	8789	9258	9739	10232	10737	11255	11784	12326	15217		
8 0	6842	7263	7697	8143	8602	9073	9557	10053	10562	11084	11618	12164	12723	16708		
8 3	7056	7490	7937	8397	8870	9356	9855	10367	10892	11430	11981	12544	13121	16199		
8 6	7270	7717	8178	8652	9139	9640	10154	10681	11222	11776	12344	12925	13519	16890		
8 9	7484	7944	8419	8906	9408	9924	10453	10996	11552	12123	12707	13305	13916	17181		

9 0	7698	8171	8659	9161	9677	10207	10751	11310	11882	12469	13070	13685	14314	17672
9 3	7912	8398	8900	9415	9946	10491	11050	11624	12212	12815	13438	14065	14712	18162
9 6	8125	8625	9140	9670	10214	10774	11349	11938	12542	13162	13796	14445	15109	18658
9 9	8339	8852	9381	9924	10488	11058	11647	12252	12872	13508	14159	14825	15507	19144
10 0	8553	9079	9621	10179	10752	11341	11946	12566	13202	13854	14522	15205	15904	19635
10 3	8767	9306	9862	10433	11021	11625	12245	12881	13533	14201	14885	15585	16302	20126
10 6	8991	9533	10102	10688	11290	11908	12543	13195	13863	14547	15248	15966	16700	20617
10 9	9194	9760	10343	10942	11559	12192	12842	13509	14193	14893	15611	16346	17097	21108
11 0	9408	9987	10583	11197	11827	12475	13140	13823	14523	15240	15974	16726	17495	21599
11 3	9622	10214	10824	11451	12096	12759	13439	14137	14853	15586	16337	17108	17892	22089
11 6	9836	10441	11064	11706	12365	13042	13738	14451	15183	15933	16700	17486	18290	22580
11 9	10050	10668	11305	11960	12634	13326	14036	14766	15513	16279	17063	17866	18688	23071
12 0	10264	10895	11545	12214	12903	13609	14355	15080	15843	16625	17426	18246	19085	23562
12 3	10477	11122	11786	12469	13171	13898	14634	15394	16173	16972	17790	18627	19483	24063
12 6	10691	11349	12026	12728	13443	14176	14932	15708	16503	17318	18153	19007	19880	24544
12 9	10905	11576	12267	12978	13709	14460	15231	16022	16833	17664	18516	19387	20278	25035
13 0	11119	11803	12507	13232	13978	14744	15530	16336	17163	18011	18879	19767	20676	25526
13 6	11547	12257	12989	13741	14515	15311	16127	16965	17823	18703	19605	20527	21471	26507
14 0	11974	12711	13470	14250	15053	15878	16724	17593	18493	19396	20331	21287	22266	27499
14 6	12402	13165	13951	14759	15591	16445	17322	18221	19144	20089	21057	22048	23061	28471
15 0	12830	13619	14432	15268	16128	17012	17919	18850	19804	20782	21783	22808	23857	29453
15 6	13257	14073	14913	15777	16666	17579	18516	19478	20464	21474	22509	23568	24652	30434
16 0	13685	14527	15394	16286	17203	18146	19113	20106	21124	22167	23235	24328	25447	31416

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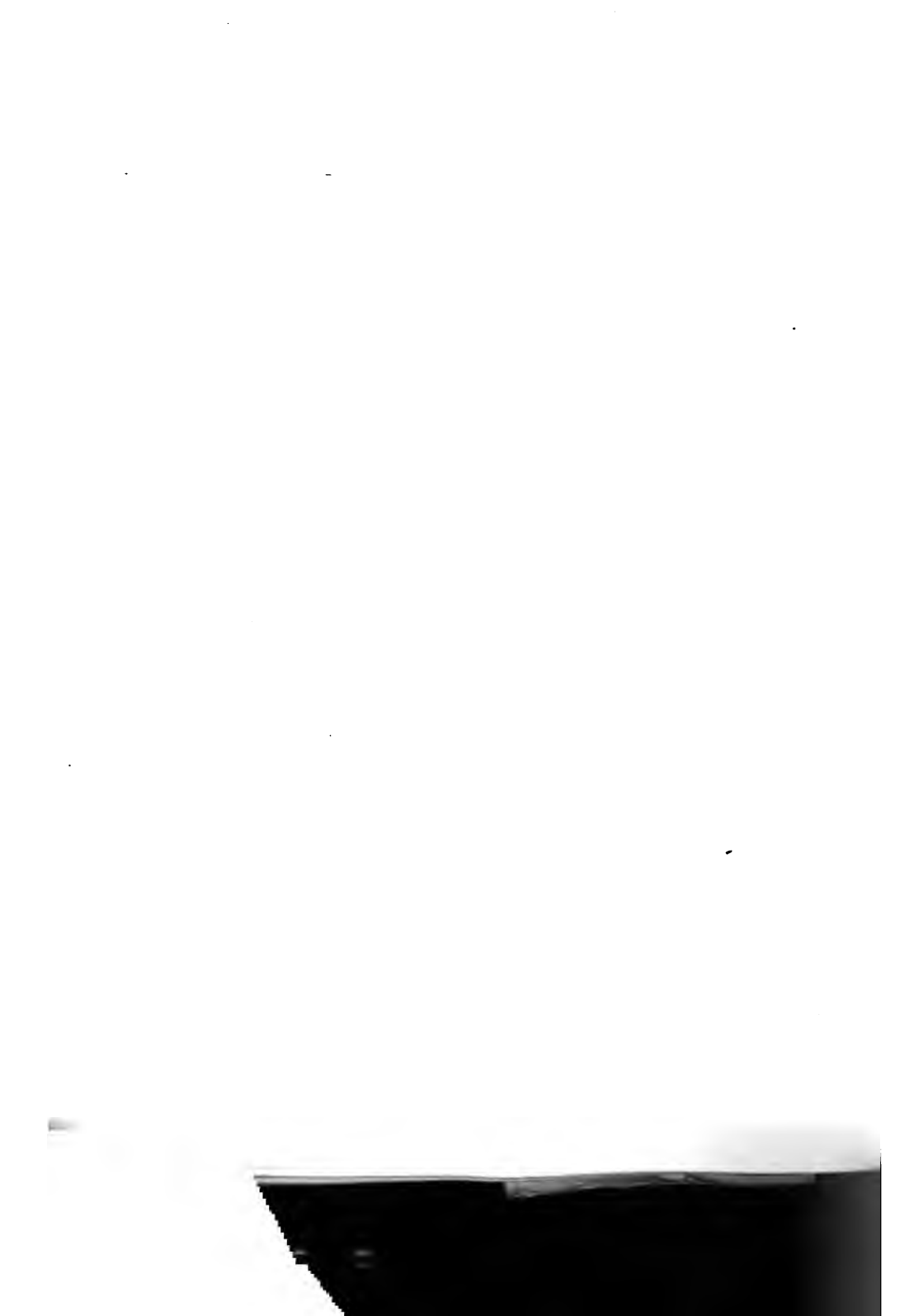
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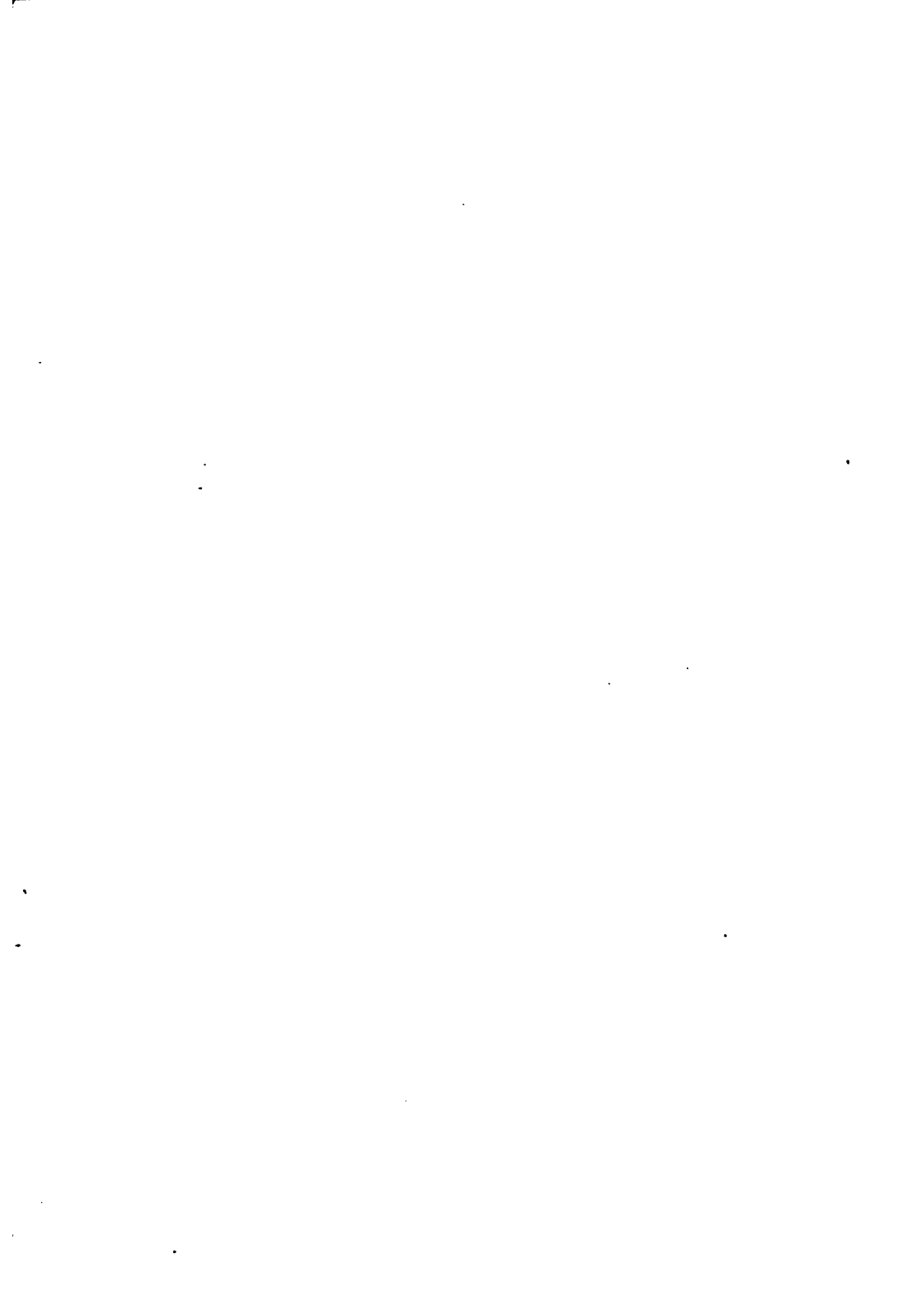














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